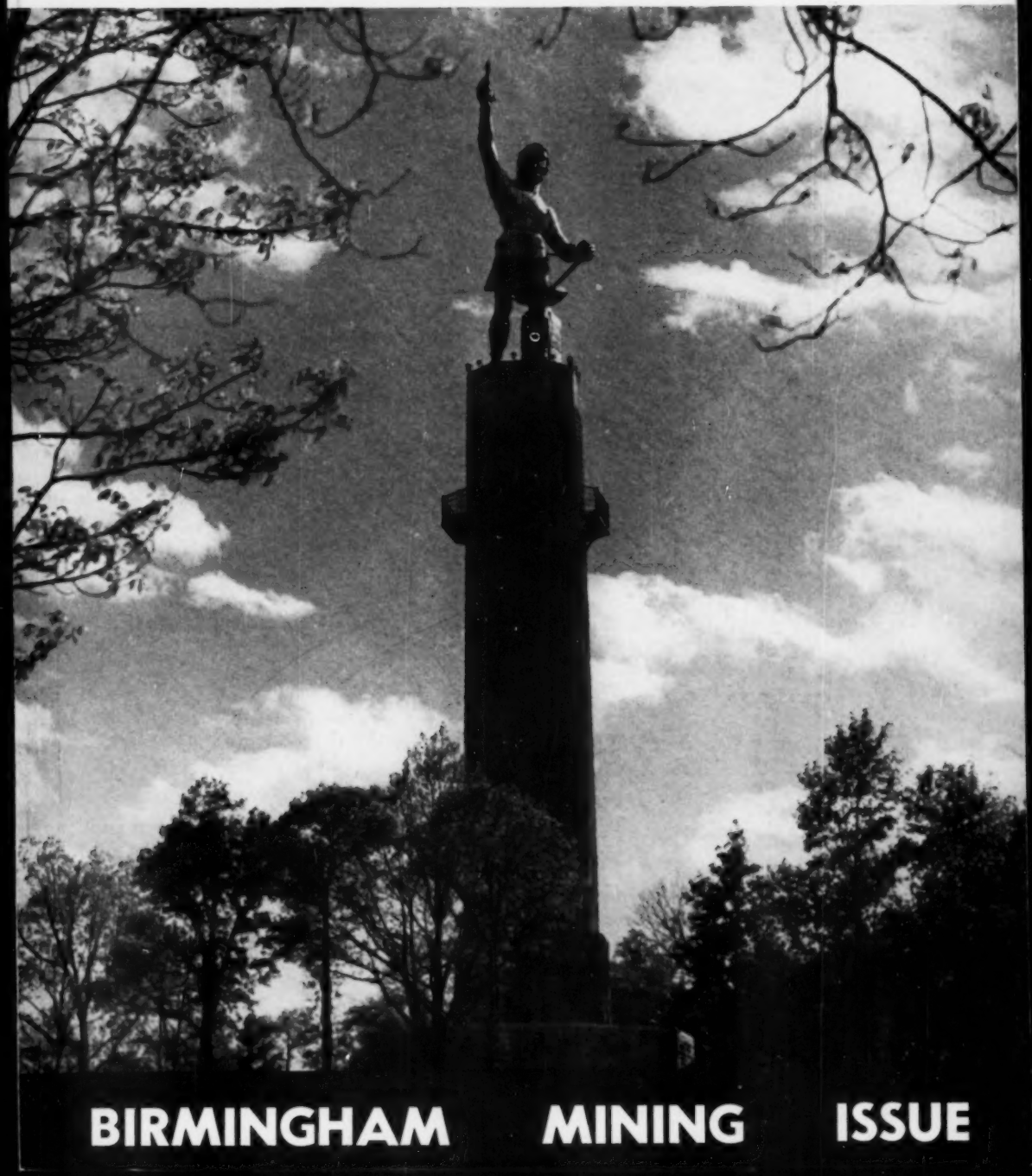
MINING ENGINEERING

DECEMBER 1950



BIRMINGHAM MINING ISSUE

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Centrifugal Sand Pump

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The new WILFLEY Model K Centrifugal Sand Pump features important mechanical improvements which give you substantial dollar savings—in power and production. This new pump delivers continuous, trouble-free operation—stepped-up production—dependable performance in handling sands, slimes or slurries. Individual engineering on every application—an economical size for every requirement. Write or wire for details.

Write for Wilfley Bulletin which details the special features, capacities, dimensions and other pertinent information.

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MINING ENGINEERING

Incorporating Mining and Metallurgy, Mining Technology and Coal Technology

VOL. 187 NO. 12 DECEMBER 1950

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COVER: Atop Red Mountain, overlooking the City of Birmingham, is Vulcan, Greek god of fire and metalworking, who was later a part of Roman mythology. He is, of course, the symbol of Birmingham's industrial might, to which this issue is dedicated.

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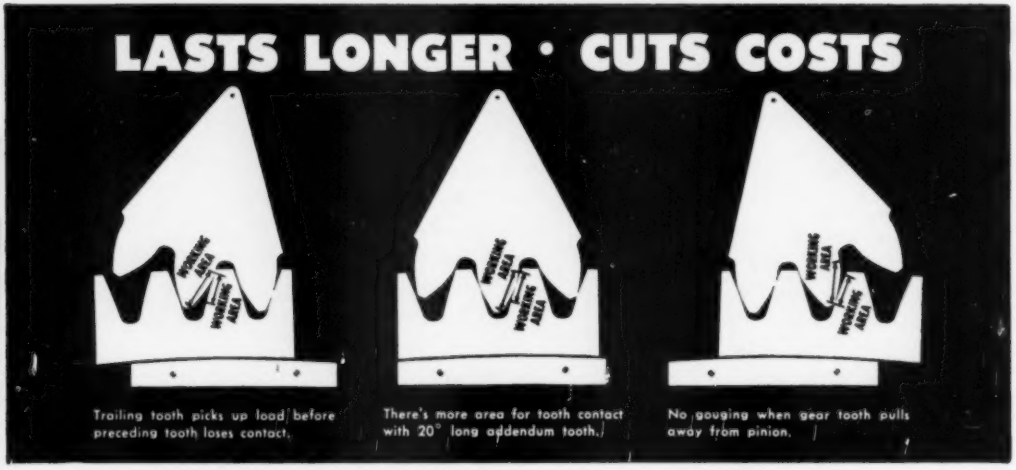
NOW! New Smoother Running Gear

LASTS LONGER • CUTS COSTS



Trailing tooth picks up load before preceding tooth loses contact.

There's more area for tooth contact with 20° long addendum tooth.

No gouging when gear tooth pulls away from pinion.

Big news for operators of grinding mills, rotary kilns, coolers, dryers! Allis-Chalmers offers new 20° involute cut tooth spur gear . . . developed after a comparative survey proved these gears the most effective for smooth operation and long gear life.

HERE'S WHY new 20° involute spur gear with long addendum pinion and short addendum gear tooth results in smooth, even distribution of force from pinion to gear:

1) Gear tooth *rolls* evenly on pinion tooth, not first rolling and then sliding. 2) Rolling action results in least friction wear. Pressure is lower because working force is distributed over 75% more tooth area. 3) Each gear tooth is 25% thicker at base, 25% stronger per inch of gear face.

Gear teeth cannot rub on following pinion tooth when it pulls away on turn. Each gear tooth is a true involute shape, cut on precision machines and rigidly inspected to assure large gears of high accuracy.

This improved gear is standard on new A-C equipment; can, in many cases, be obtained to replace existing gears without changing center to center distances. For more facts, call the Allis-Chalmers man in your area.

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SULPHUR

*Interesting Facts Concerning This Basic Raw Material from the Gulf Coast Region

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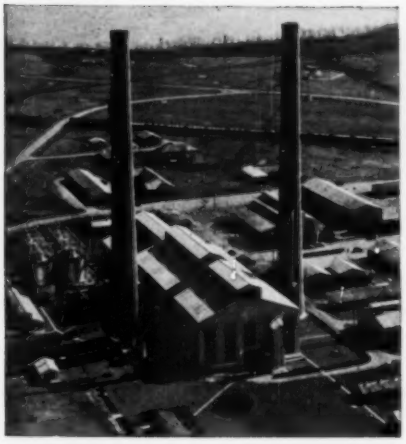

The process of mining sulphur, as developed by Herman Frasch, takes advantage of the fairly low melting point of sulphur (about 240° Fahrenheit). The process resolves itself into three parts: one, operating a power plant that heats and pumps to the field large quantities of water; two, distributing the hot water through wells to melt the underground sulphur, and raising the melted sulphur to the surface; three, cooling and solidifying the sulphur in large vats from which it is broken and loaded into cars for shipment.

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IN THIS 16-PAGE BOOKLET

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Introduction 1
The Flotation Process 2
Flotation Agents and Their Application . . . 2
Frothers 2
Collectors 2
Modifiers 3
Quantity of Agent 3
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Conditioning 4
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Cellulosics—as Depressants, Flocculators, and Dispersers 6
Appendix 6

This new helpful booklet "Flotation and Hercules Flotation Agents," is designed for mining and metallurgical engineers and for all others who are interested in the economic beneficiation of ore beneficiation.

Included in this 16-page booklet are descriptions of the flotation process and the functions of flotation agents, as well as information about Hercules' frothers, collectors, and modifiers. A special section of mathematical formulas useful in flotation is of added practical value.

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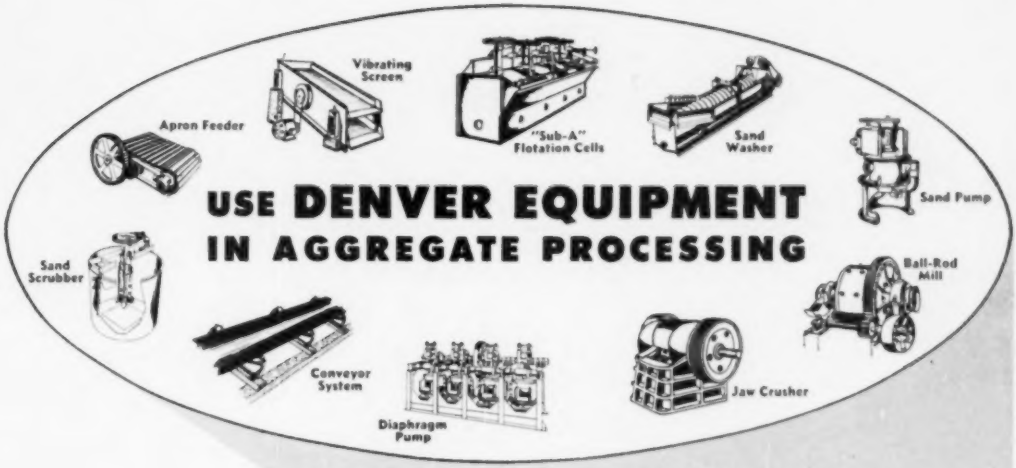

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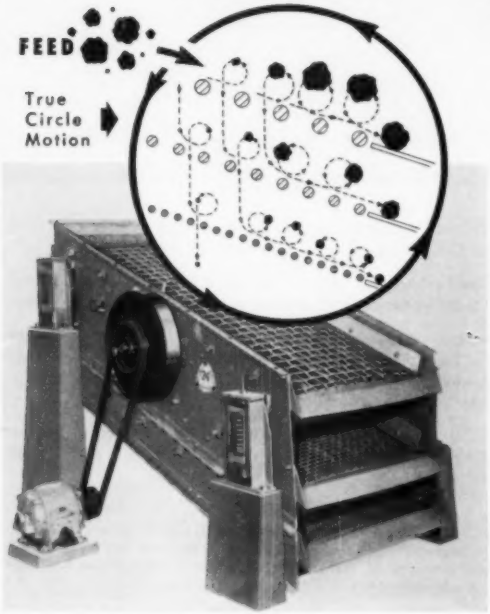

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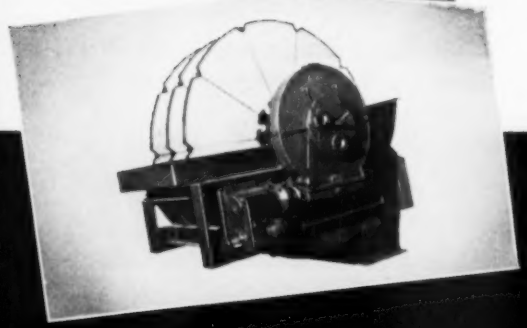

EIMCO EQUIPMENT IN THE BIRMINGHAM DISTRICT

In the Birmingham area, as in every other mining area, Eimco machines are doing an important job.

RockerShovels—both underground and on the surface are loading rock and ore cheaper and reducing costs.

Filters — Eimco Continuous Vacuum Filters in both drum and disc types are dewatering concentrates, flue dust and other products. They give more capacity per square foot with lower moisture.

A dependable performance record for over half a century has enabled users to rely on Eimco and to all agree — you can't beat an Eimco.

(Above) Eimco Model 40 loading iron ore. (Left top) Large cars are loaded full off Model 40 conveyor. (Middle) Crawler mounted 102 RockerShovels working underground. (Bottom) Eimco Continuous Vacuum Disc filters are used in several Birmingham plants.

A227

EIMCO

THE EIMCO CORPORATION

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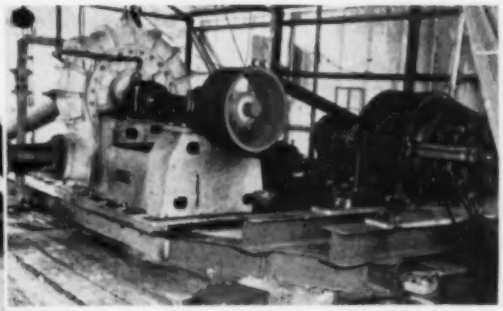
"We PROFITED by Converting to Thomas Pumps"

Says A. G. Collins, Manager, American Sand & Gravel Company, Hattiesburg, Mississippi.



Above: Twelve-inch Thomas Durable Pump operating on 12-inch pipe line shown at left.

Left: Foreground shows 12-inch pipe line from Thomas Durable Pump to screening plant on shore. In front of screening plant, is 8-inch Thomas Pump serving loading plant of American Sand & Gravel Company, Hattiesburg, Mississippi.

MODERN DESIGN PLUS MODERN MATERIALS EQUAL LOWER PUMPING COSTS

Mr. Collins Writes:

"Our 8-inch Thomas pump, which replaced a cast iron pump 14 months ago, loads about one-third more material than we did previously in a given time, our power unit is not overloaded as it was before, and Thomas pump parts outlast those in our previous pumps by 5 or 6 times."

★ ★ ★ ★

"Our 12-inch Thomas pump replaced a manganese steel pump and gives us 3 to 4 times more wear than the unit replaced. Our downtime has decreased to almost nothing."

★ ★ ★ ★

"We are now installing a 6-inch Thomas pump to replace our last manganese steel pump. We have profited considerably by converting our entire operation to Thomas pumps."

This Thomas Durable Pump user is making a "clean sweep" of his old pump equipment. Within about one year, he has replaced three ordinary, conventional pumps with Thomas Durable Pumps.

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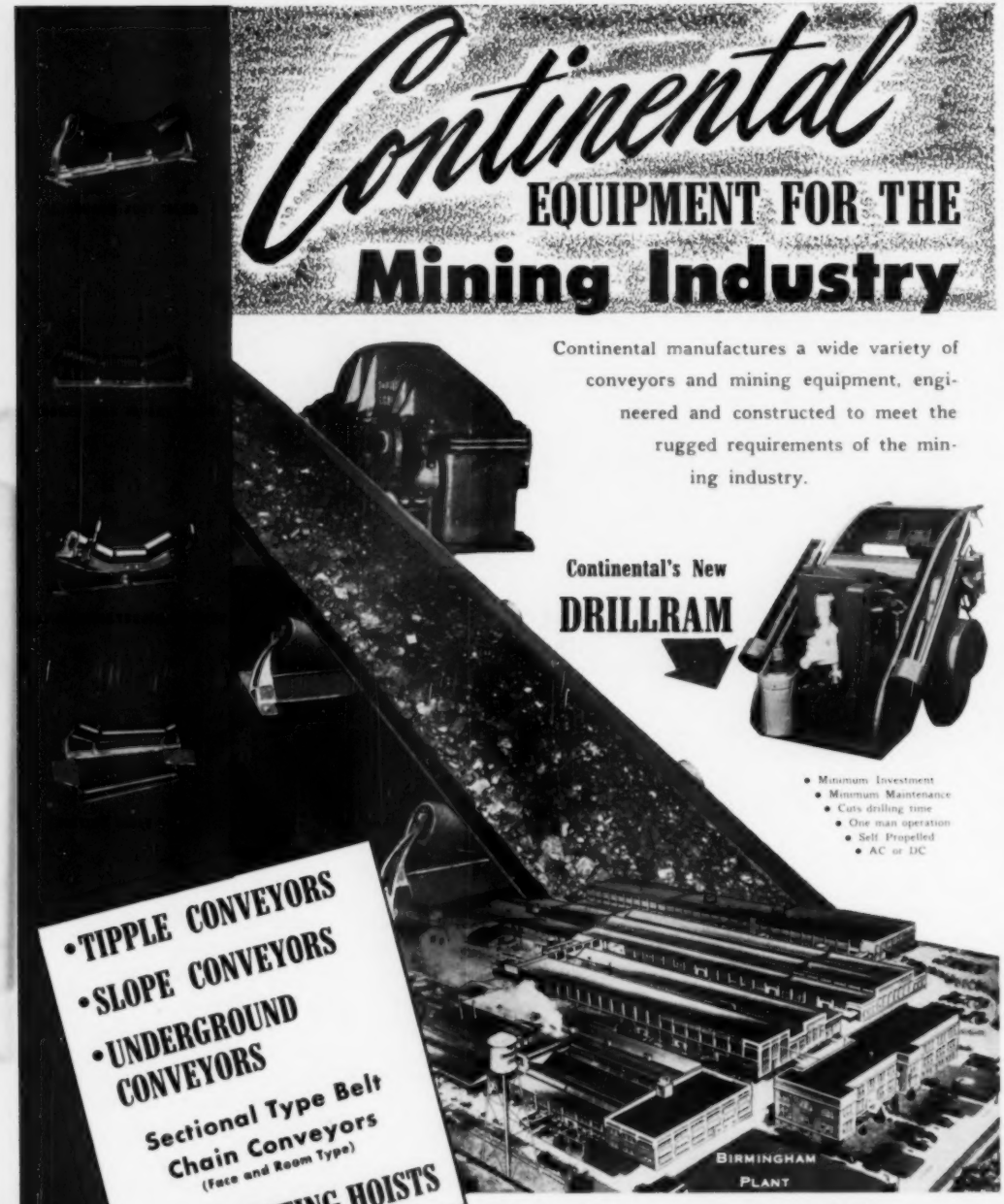

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(Face and Room Type)

- CAR SPOTTING HOISTS

BIRMINGHAM PLANT

Write for Belt Conveyor Data Book ID-481-A.

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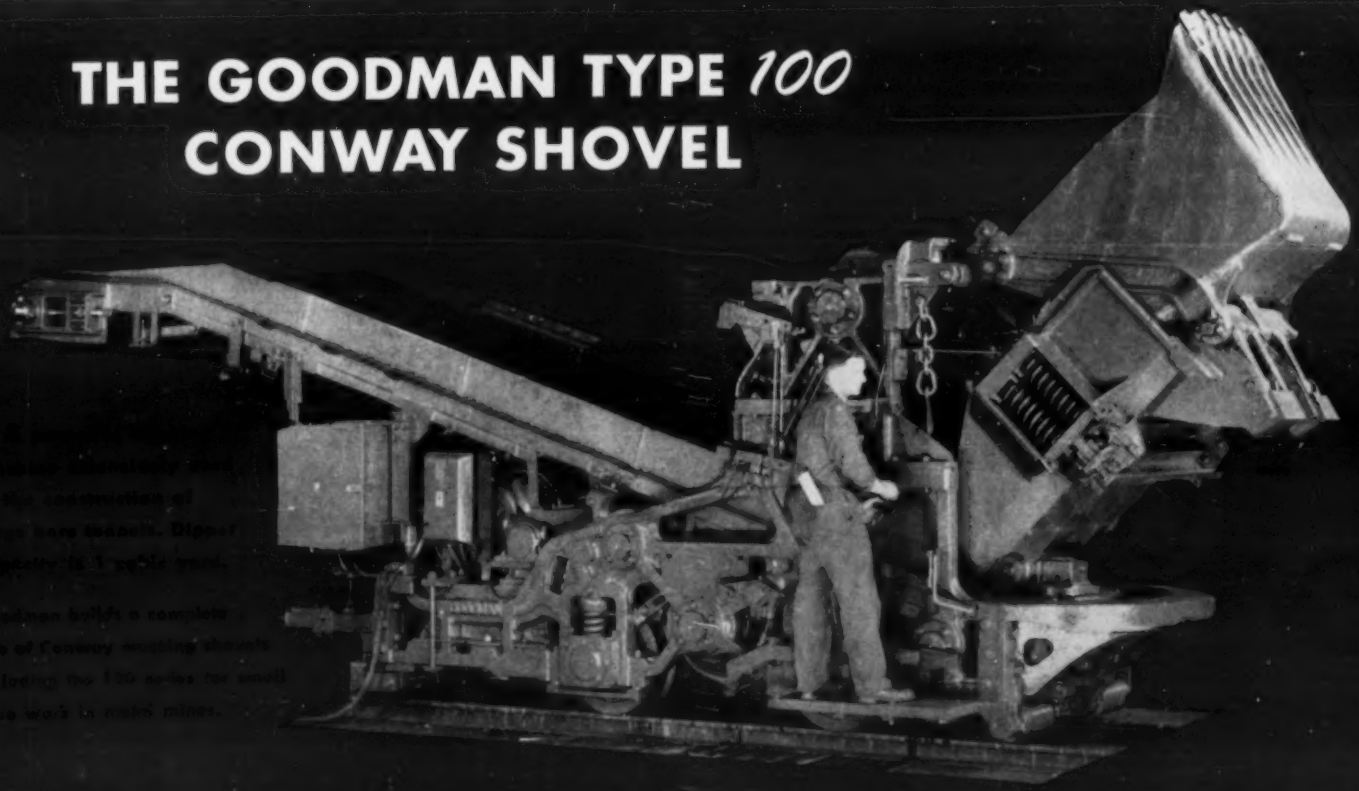

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GOODMAN MANUFACTURING COMPANY

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4 NORDBERG MINE HOISTS

Serve Major Birmingham Mining Company

IN the past 14 years, four Nordberg Mine Hoists have been installed in the Birmingham, Ala., district by the South's largest coal and iron producer. One of these, illustrated above, is an 18′ diameter, 17′-6″ face, single drum hoist . . . believed to be the largest of its kind in a slope mine operation. It operates on a 6700 foot incline, which varies up to a maximum of 25 degrees.

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HOIST DATA:

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Maximum hoisting	3600 feet per minute
Maximum lowering	3300 feet per minute
Weight of Ore	35,000 pounds
Weight of Skip	23,000 pounds
Motor*	3500 H.P. A.C.

*This motor is one of the largest A.C. motors ever installed for mine service.

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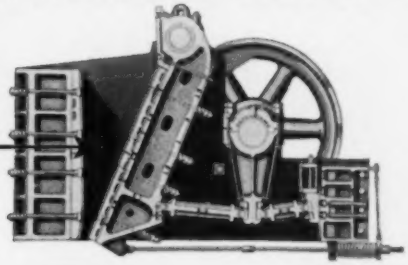
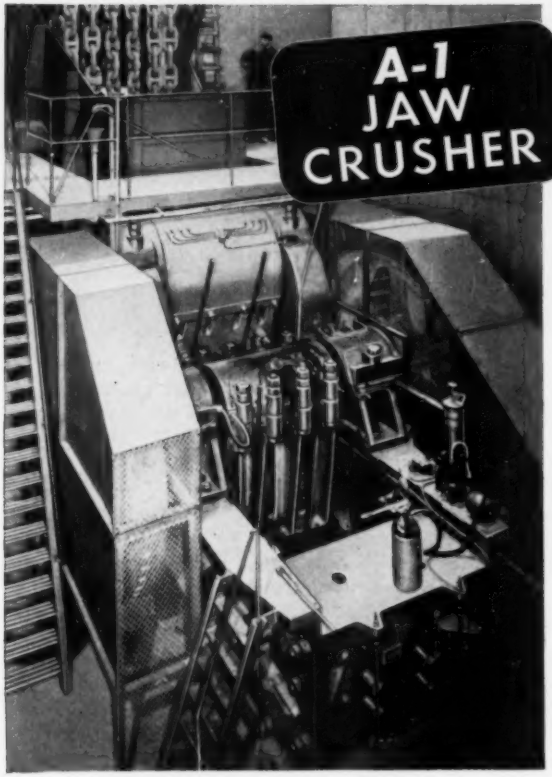
HERE'S THE JAW CRUSHER WITH THE

DEEP



Crushing Chamber



60 x 48 A-1 Jaw Crusher Installed Abroad.

THIS IS WHAT *A-1* jaw crusher's long, deep crushing chamber (up to 35% deeper than conventional crushers) means to you in terms of crushing performance and economy:

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Uniform wear on jaw plates — curved plates distribute crushing action over greater surface, resulting in better economy.

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- ★ Automatic lubrication of main, pitman and swing jaw bearings.
- ★ Jaw plates suspended from wearing plates, permitting free expansion of manganese steel.
- ★ Sectionalized construction permits ease of handling and installation.

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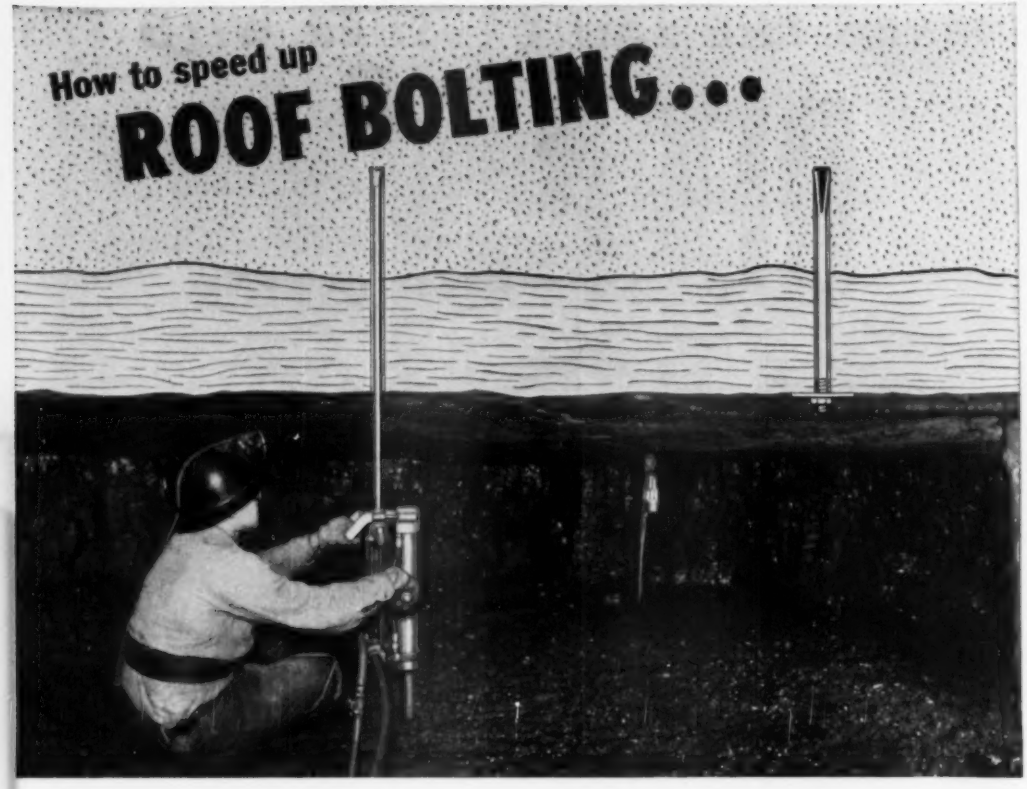

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more power, less weight

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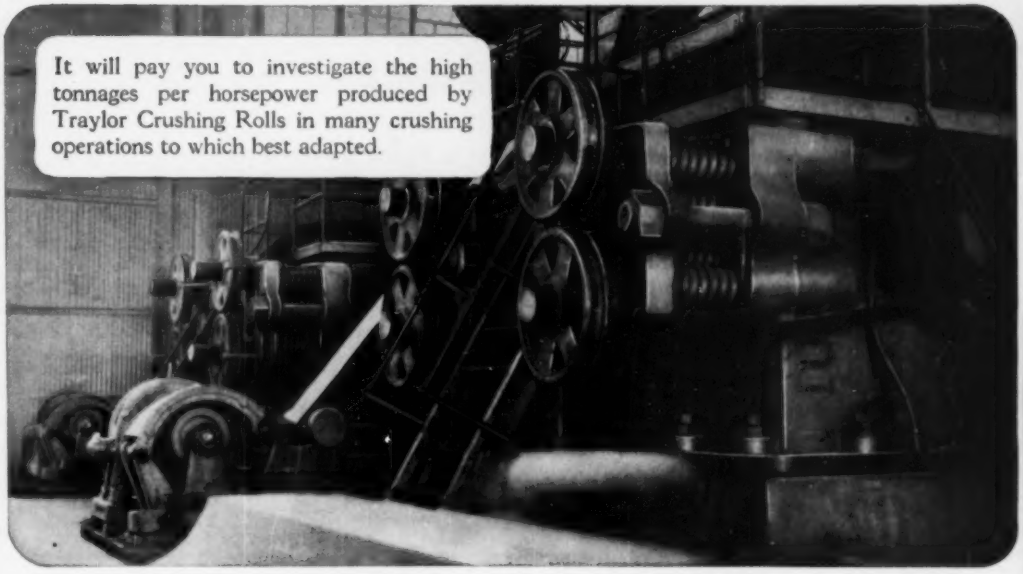
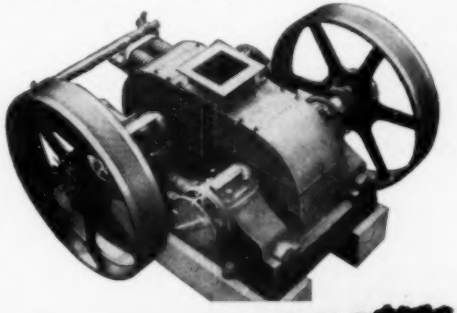
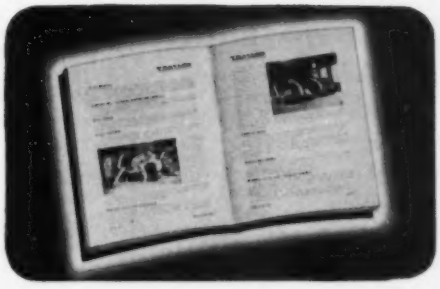


It will pay you to investigate the high tonnages per horsepower produced by Traylor Crushing Rolls in many crushing operations to which best adapted.

Here are actual performance figures - - Traylor 78" x 20" Rolls (main illustration) operated by a large mining company, consistently crush primary feed of 310 t.p.h. plus 180% circulating load to 52% minus 8 mesh. And have done so for a long time. A pair of smaller rolls, 36" x 16" and 42" x 16" consistently runs 125-150 t.p.h. to ⅛"-3/16". They then reduce oversize, which has been screened out, to 6 mesh at a rate of 120 t.p.h.





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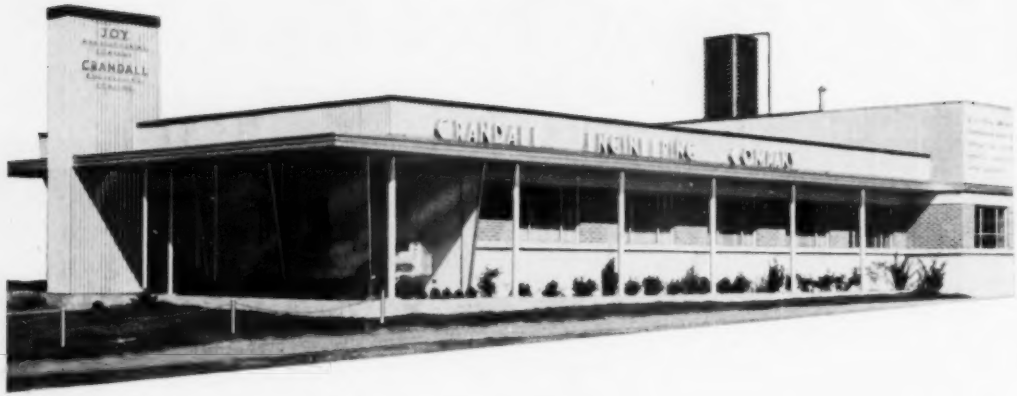

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THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service, Inc., operating in cooperation with the four Founder Societies. Local offices of the Personnel Services are at 8 W. 40th St., **New York 18**; 100 Farnsworth Ave., **Detroit**; 57 Post St., **San Francisco**; 84 E. Randolph St., **Chicago 1.** Applicants should address all mail to the proper key numbers in care of the New York Office and include 6¢ in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter or $12 a year.

MEN AVAILABLE

Dutch Graduate Mining Engineer, 58, married, desires position as general manager. Under his leadership have been built several mines in primitive countries with coal and ore handling—and dressing plants, with all kinds of installations. Speaks five languages. Available immediately. M-598.

POSITIONS OPEN

Instructor to teach mineral prospecting, mine surveying and structural mining geology. Must be able to carry on research work. Salary open. Location, East. Y4467.

Engineers. (a) Materials handling engineer with experience on coal preparation, to survey and plan improvements for modernizing coal cleaning plant. Salary, up to $7500 a year. (b) Mechanical or mining engineer with coal preparation experience, to design and lay out structures and equipment for coal cleaning plant. Salary, $3600-$4800 a year. Location, western Pennsylvania. Y4365.

Sales Engineer to sell mining machinery in the eastern district. Must have previous experience in mining, coal preparation, etc. Salary, $3600 a year to start plus bonus. Location, New York, N. Y. Y3968.

Mining Engineer experienced in using latest technical means to develop and work out manganese mines. Should be familiar with the machineries and from where those can be procured. Appointment will be made for three years. Traveling expenses paid. Location, India. Y3949.

Assistant Chief Engineer, graduate, with several years' experience in the coal industry, and who is particularly interested in engineering, not operating, for underground mine engineering and surface construction work. Prefer married men. Location, Pennsylvania. Y3936.

WANTED: Immediately for base metal mine, Bolivia, Chief Engineer. Some knowledge Spanish advisable but not essential. Would consider recent graduate with some background operating experience. Standard three-year contract, travelling expenses both ways. Starting salary $3300—annually up depending on experience.

Box K-22 MINING ENGINEERING

AVAILABLE

METALLURGICAL ENGINEER, 38 years old. Experienced in gold refinery, ore dressing, electro chemistry and research. Graduate of Freiberg, Saxony. Languages: English, German, Russian.

Box K-23 MINING ENGINEERING

Costs Less per Service Year

SIMPLEX
CAST IRON PIPE FOR COAL MINES

EXPERIENCED operators know that the true cost of mine pipe includes more than first cost: it includes installation and upkeep expense, all lumped together and spread over the line's period of service.

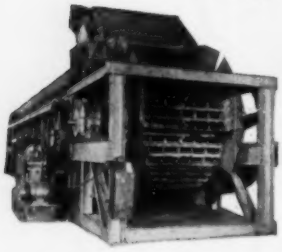
Simplex Pipe costs less per service year because the first cost compares favorably with that of inferior pipe, while installation and upkeep expense is lower and the period of service infinitely longer!

Simplex cast iron pipe outlasts most coal mines, but it's easy to take up and move to a new mine. It is manufactured in convenient five-foot lengths with bolted joints that enable you to take up your old Simplex lines and move them to new locations, easily and economically.

AMERICAN CAST IRON PIPE COMPANY

BIRMINGHAM, ALABAMA

AORCO



SINTERING MACHINES

HAVE PROVED THEIR RELIABILITY

For continuous operation with the minimum of maintenance and the lowest cost per ton of sinter. They are sturdily built to resist heat, shock and corrosion. Sizes range up to 1500 square feet of hearth area for single units.

Auxiliary equipment built by AORCO includes feeders, mixers, fans, dust collectors and sinter coolers. AORCO engineering service covers design and construction of complete plants.

AMERICAN ORE RECLAMATION CO.
310 S. MICHIGAN AVE. CHICAGO 4, ILL.

BUILDINGS EQUIPMENT MACHINERY

GIGANTIC

LIQUIDATION

CRUSHERS BALL MILLS GENERATORS ROLLING MILLS MOTORS ETC

Formerly American Smelting & Refining Co. MURRAY PLANT

AVAILABLE NOW IMMEDIATE SALE SACRIFICE THESE ITEMS MUST BE SOLD AT A FRACTION OF ORIGINAL COST IMMEDIATE DELIVERY

BUILDINGS AND STRUCTURAL STEEL
30—Assorted Steel Construction Buildings mostly with corrugated sidings and roofs. These buildings may easily be removed and re-erected on your site
12,000,000 Used Brick
Large quantity of Galvanized Iron

CRUSHERS, ROLLS AND BALL MILLS
7—Assorted Crushers, Rolls and Ball Mills

ELEVATORS AND CONVEYORS
15—Assrd. Bucket Elevators, up to 170 ft
25—Assrd. Belt Conveyors up to 130 ft

FEEDERS
25—Assorted Jeffories Feeders, 32" x 8 and 36" x 7'

MACHINERY, ETC.
1—Engine Lathe, 36" x 16'" American Tool Co.
1—American Planer, 28" x 10'28"
1—Jarecki Pipe Threader
1—Oxweld Acetylene Generator
14—Steel Cylindrical Storage Bins, 25' x 8'6", ½" plate bolted construction 50 ton capacity
1—Steel Horizontal Tank, 8,000 gallon capacity
5—Stedman Disintegrators, 36"
7—D & L Sintering Machines, complete. 42" x 22'
8—Assorted Hershoff and Godfrey Roasters, complete

1—Wullstein Gravitation Screen, 2' x 6'.
1,000 ft.—Steel Ballast Flues, 4' x 6'.
Track, Truck, Counter and Platform Scales, Electric Freight Elevators

MOTOR GENERATOR SETS
12—Westinghouse Motor Generators, 15 KVA with 20 HP motors.
2—G. E. Motor Generators, 50 KW with 50-75 HP motors
2—Westinghouse Generator Exiter Sets

PUMPS AND BLOWERS
20—Worthington, Etc., Triplex and Centrifugal Pumps, 300-2,000 GPM
10—Assorted Roots-Connerville, Etc Blowers

ROLLING STOCK, RAIL AND LOCOMOTIVES
1—Davenport 10 Ton Diesel Locomotive, 36 Gauge, D8800 Caterpillar Power Unit.
15—Standard Steel Semi-Dump Cars
58—Assorted Mine and Mill Ore Cars
1—Bay City Standard Gauge Locomotive Crane
17—Solid Slag Pots, cast steel frames
2—H. R. Porter 10 Ton Steam Locomotives
600 Tons—Rail, 40-75 lb. and smaller
2—Charge Spreading Apparatus
Tractors and Trucks. Fish Plates, Spikes, Frogs, Switches, Ties, Etc

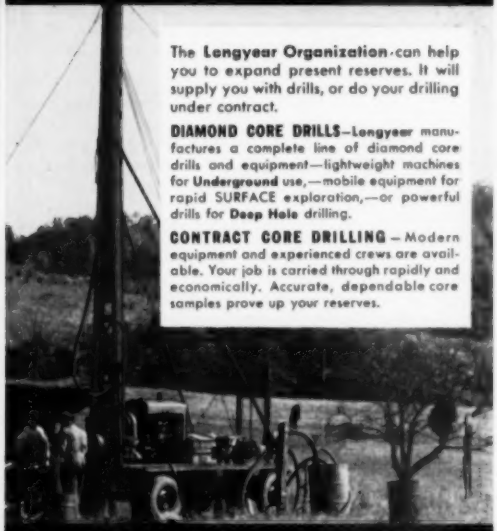
TRANSFORMERS
3—Transformers, 833 KVA
75—Transformers, 5-500 KVA

MISCELLANEOUS ITEMS
Office Equipment, Steel Lockers, Hand Tools, Machine Tools, Gears, Pulleys, Belting, Supplies, Etc. Etc.

ALL ABOVE SUBJECT TO PRIOR SALE

WRITE · WIRE · PHONE for further information and COMPLETE listings
MURRAY-AMERICAN MILL • 5250 SO. STATE ST. MURRAY UTAH
LIQUIDATORS IN CHARGE of former AMERICAN SMELTING & REFINING CO. MURRAY PLANT
★PLANT → LOCATED on HIGHWAY 91, with ACCESS RAILS to UP and D&RG TRACKS • 7 MILES from SALT LAKE

INCREASE MINERAL RESERVES BY DIAMOND CORE DRILLING



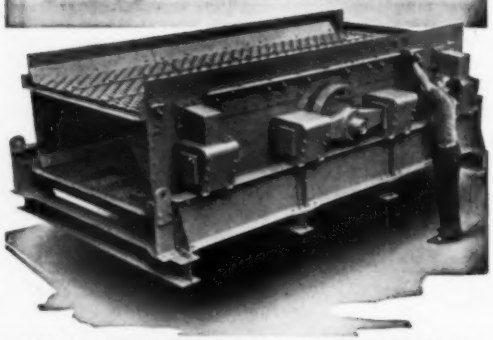
The **Longyear Organization** can help you to expand present reserves. It will supply you with drills, or do your drilling under contract.

DIAMOND CORE DRILLS—Longyear manufactures a complete line of diamond core drills and equipment—lightweight machines for **Underground** use,—mobile equipment for rapid SURFACE exploration,—or powerful drills for **Deep Hole** drilling.

CONTRACT CORE DRILLING—Modern equipment and experienced crews are available. Your job is carried through rapidly and economically. Accurate, dependable core samples prove up your reserves.

E. J. LONGYEAR COMPANY
MINNEAPOLIS, MINNESOTA, U. S. A.
DIAMOND CORE DRILLS • CONTRACT CORE DRILLING • SHAFT SINKING
GEOLOGICAL INVESTIGATIONS

TY-ROCK — the Ideal Screen for HEAVY LOADS—COARSE MATERIALS!



USED THROUGHOUT THE WORLD WHEREVER LARGE TONNAGES OF ROCK OR ORE ARE SCREENED!

Manufacturers of Woven Wire Screens and Screening Machinery

THE W. S. TYLER COMPANY
CLEVELAND 14, OHIO • U. S. A.

Dear Editor

IN the October, 1950 issue under "Reporter" I note the following:

"Great Britain's socialized coal industry employs twice as many miners, but produces less than half as much coal as America's coal industry."

This is a repetition of a statement that one sees so often in publications in this country which I submit has little if any relevance and usually is interpreted to mean something, that most of those who know, would insist is not a provable fact.

Actually is it not true that the physical conditions are in no way comparable.

Would it not be just as valid to criticize the anthracite industry of Pennsylvania because its output per man shift is so much less than that of the bituminous industry of this country?

A further set of facts which may or may not have relevance is that the percentage of extraction of bituminous coal in the ground in Great Britain is far greater than shown by the bituminous mines of this country. However, one seldom encounters any reference to this comparison.

My comment is not made with the desire to defend socialized industry. However, I contend that only the last two sentences in the paragraph are valid criticisms of Great Britain's socialized coal industry.

John Griffen
McNally Pittsburgh Mfg. Co.

Hoover's

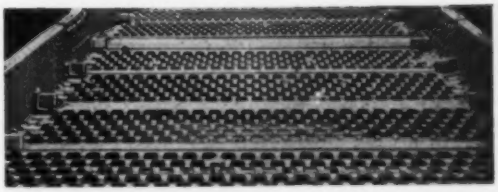
"De Re Metallica"
Reissued

Collectors have paid up to $150 for the famed Herbert Hoover Translation of Agricola's timeless work. The volume has been republished, and a *limited number* of copies are available at only $10. 672 pages, 289 illustrations. FIRST COME, FIRST SERVED.

AIME
29 West 39th St. check enclosed .....
New York 18, N. Y. bill me at this address .....

Please send me ..... copies of "De Re Metallica", at $10 each:

Name .....
Company .....
Street .....
City, Zone, State .....
.....



148 SLOT SIZES
in Flanged Lip Screens
meet exacting requirements

Originally developed as a superior medium for the screening of coal, Hendrick Flanged Lip Screens today are also being extensively used for screening ore, stone, gravel, and other granular materials.

The staggered slots are practically non-clogging, and the step at the end of each slot aids in producing faster and cleaner separation of the material. Made with short, medium or long slots; 148 different standard slot sizes are available. Write for a copy of Flanged Lip Screen booklet.

Perforated Metals
Perforated Metal Screens
Architectural Grilles
Mitco Open Steel Flooring,
"Shur-Site" Treads and
Armorgrids

HENDRICK
Manufacturing Company
44 DUNDAFF STREET, CARBONDALE, PA.
Sales Offices in Principal Cities

SAUERMAN

Power Drag
Scrapers

Picture shows Sauerman Drag Scraper stacking out crushed fines on a pile 550 ft. in radius.

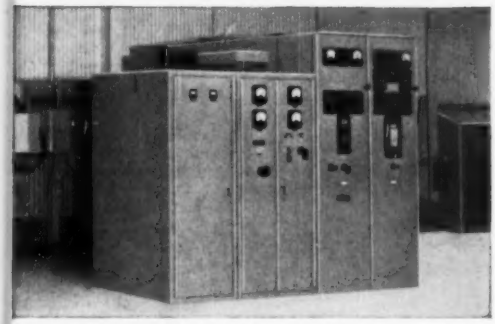
Whatever your material — ore, coal, boulders, sand, gravel, clay—you can move it fast and efficiently with a long range Sauerman machine of either cableway or scraper type—at a great saving in cost. Any mechanic can operate it. Upkeep is simple. Machine is flexible; range is easily extended. Sturdy design and construction for long service. Bucket sizes run from 1/3 to 15 cu. yd.; spans range up to 1,000 ft.

WRITE TODAY FOR LATEST CATALOG

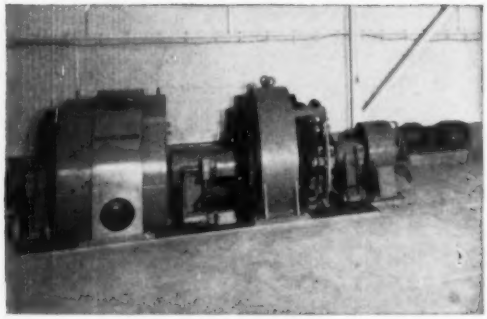
SAUERMAN BROS., INC. 546 S. CLINTON ST. CHICAGO 7, ILL.

Small Sauerman Scraper Handles 30,000-Ton Stockpile, Storing and Reclaiming 400 tons a day.

AUTOMATIC HOISTING for continuous tonnage!



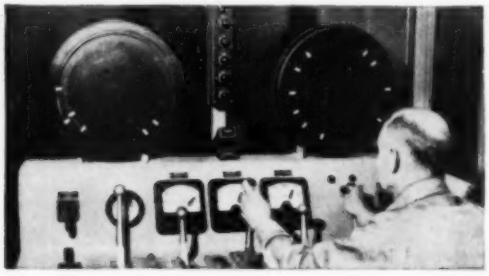
Completely metal-enclosed for personnel safety, this control cubicle houses the synchronous motor switchgear panel, plus Ward Leonard type d-c control equipment. Less space requirement and simplified control result from use of the G-E amplidyne, which eliminates need for many contactors and relays, while permitting the retention of safe limits on the hoist's acceleration and deceleration.



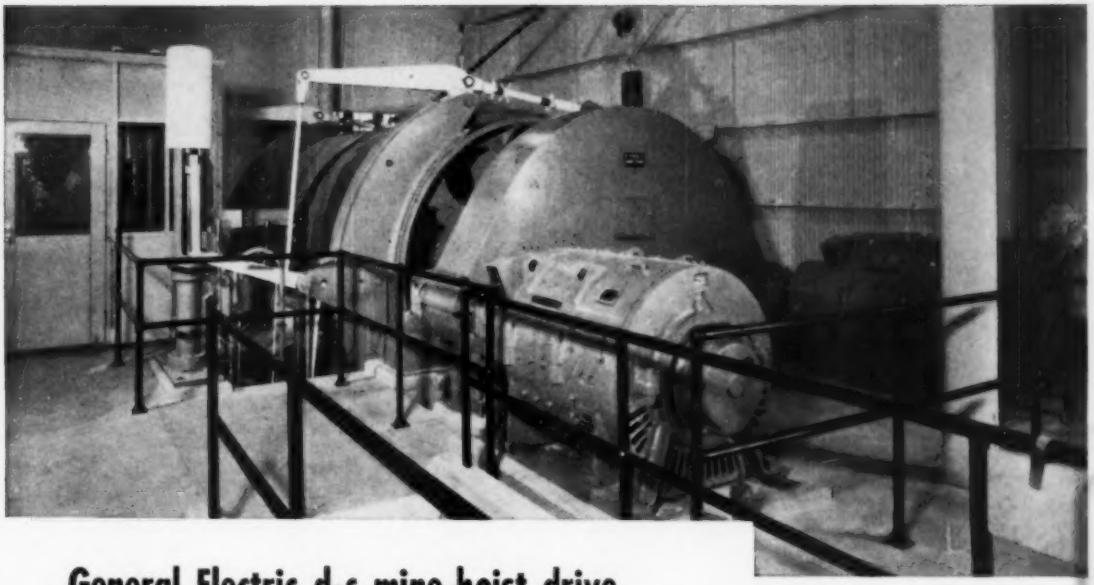
Converting a-c to d-c, this G-E motor-generator set comprises a 750-kw generator, an 1100-hp synchronous motor, and a 20-kw exciter. The generator supplies adjustable-voltage power for the operation of the two hoist motors. At right, the G-E amplidyne, used as an exciter for this generator, provides superior operation and more positive limits of acceleration and deceleration, for greater safety, increased production.

MINE-HOIST DRIVES

—to cut mining costs!



When necessary for inspection, testing, or man-trips, control of the hoist can quickly be switched from automatic to the manually operated master controller as above. Adequate starting, stopping, and maneuvering of the skips with ease, precision, and safety is provided by the optional manual operation. With the normal automatic operation, the duties of the operator may be chiefly of another nature.



This automatic mine hoist is powered by a complete G-E drive that includes two 500-hp d-c motors. When the skip at the shaft bottom is fully loaded, it starts and accelerates to full speed. Near the dumping pocket, it automatically slows down, eases through the dumping horns, and makes a dead stop at the final limit of travel, thus completing the entire cycle automatically.

General Electric d-c mine-hoist drive with amplidyne control permits continuous output at 8 tons per minute, simplifies operation, increases safety, prolongs equipment life!

Here is another General Electric amplidyne-controlled d-c mine hoist drive helping to boost output and cut costs—in this case in a New Mexico potash mine.

This high-speed high-tonnage hoist raises in balance an 8-ton payload every minute up an 1150-foot shaft at speeds up to 1500 fpm. The hoist runs fully automatically throughout the entire shift, and may also be run by push-button control operated by the skip loader at the loading level.

Because it is completely automatic, this continuous, highly efficient operation permits hoisting maximum tonnage per man-hour. Hoist speed is closely maintained in spite of load variations, repetitive operation helps cut costs per ton, and personnel may be freed for other duties. Electric braking provides greater safety and prolongs the life of the hoist mechanism.

Applicable for any type of mining, this installation is further evidence of G-E experience in engineering automatic mine-hoist drives, experience that dates back to 1915. Let a G-E mining specialist put this experience to work on *your* mine-hoist problems. Call him at your nearest G-E office. Meanwhile, send for Bulletin GET-1430. *Apparatus Dept., General Electric Company, Schenectady 5, N. Y.*

GENERAL ELECTRIC

660-22

MECHANIZATION in BIRMINGHAM MINES

(Just as in mining areas <u>everywhere</u>)

IS BUILT AROUND EFFICIENT, FIELD-PROVED

JOY LOADERS, SHUTTLE CARS and DRILLMOBILES

When mining engineers in the Birmingham district saw the need for mechanized methods to decrease costs and increase profits, they turned to Joy for their equipment answers. The Joy 18-HR-1 Loader and 60-E Shuttle Car, hard-rock versions of Joy's successful coal-loading teams, furnished the necessary speed and capacity for high-production loading and haulage.

A later refinement of the loader, the Joy 18-HR-2, is now being used extensively in the area. Its 12-ton-per-hour maximum capacity permits the fastest possible movement of ore from the face. Like the loader, the Joy 60-E Shuttle Car is specially designed for the rugged duty of ore and rock handling with a minimum of maintenance.

Another machine helping to reduce costs in Birmingham mines and other mining areas is the rubber-tired Joy Drillmobile. It moves in and out fast, sets up quickly, assures drilling your rounds in minimum time and at least cost per foot of hole. ● Let us help you apply Joy Equipment to *your* conditions.

Have You seen this film?

"HARD ROCK MECHANIZED MINING"

in Sound and Full Color

★

Available for Free Showings on Request

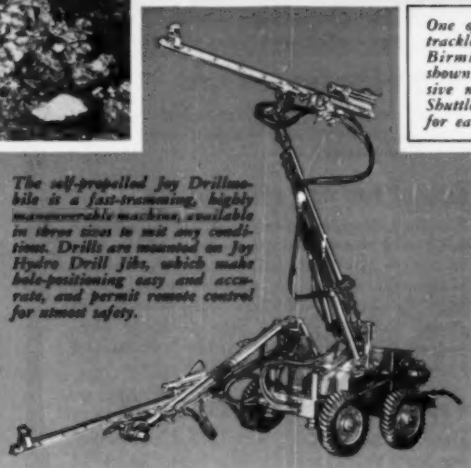
Address: Film Booking Dept.

JOY MANUFACTURING COMPANY

Oliver Bldg., Pittsburgh 22, Pa.

World's Largest Manufacturers of Underground Mining Equipment

One of the Joy continuous-type trackless loaders operating in the Birmingham Mining district, shown above loading hard, abrasive material into a Joy 60-E Shuttle Car. Controls are grouped for easy, one-man operation.



The self-propelled Joy Drillmobile is a fast-tramming, highly maneuverable machine, available in three sizes to suit any conditions. Drills are mounted on Joy Hydro Drill Jibs, which make hole-positioning easy and accurate, and permit remote control for utmost safety.

The 60-E, below, is a cable-reel operated car, easily handled and highly maneuverable.

Other JOY EQUIPMENT used in the BIRMINGHAM DISTRICT

STATIONARY AND SEMI-PORTABLE AIR COMPRESSORS—Models to deliver from 1.6 to 3656 C.F.M.

MINE-AIR COMPRESSORS—Self-propelled or draw-bar models in capacities from 130 to 240 C.F.M.

ROOF BOLTING STOPERS—Specially designed for roof bolting under most mining conditions.

CONTINUOUS MINERS—For efficient, low-cost mining in high or low coal.

TRACKLESS COAL CUTTING MACHINES—Rubber-tired, self-propelled universal cutters for all needs.

LOADING MACHINES FOR COAL—High capacity continuous-type loaders in models for all seam thicknesses.

SHUTTLE CARS FOR COAL—Easily-handled, highly maneuverable units for high capacity haulage.

CHAIN CONVEYORS—Compact and efficient, designed for long and continuous service with minimum maintenance.

SHAKER CONVEYORS—Compact sections with "Cushion Stroke" drive to handle any room and entry conditions.

Consult a Joy Engineer

WRITE FOR COMPLETE INFORMATION ON ANY OF THE ABOVE EQUIPMENT

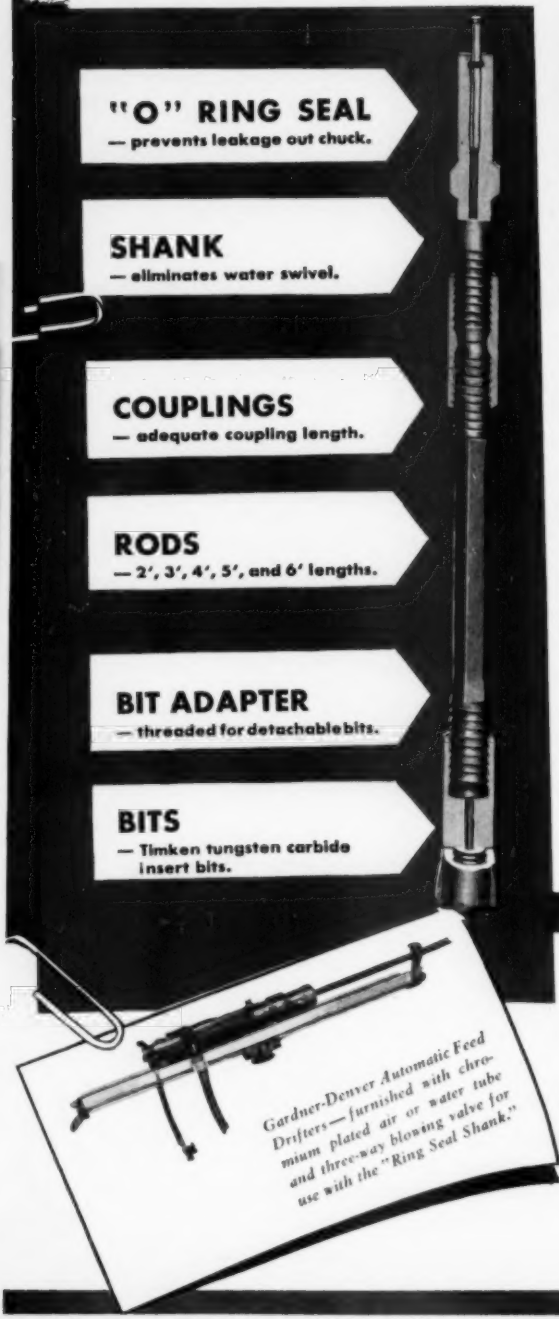
W&D M3365

JOY MANUFACTURING COMPANY

GENERAL OFFICES: HENRY W. OLIVER BUILDING · PITTSBURGH 22, PA.

IN CANADA: JOY MANUFACTURING COMPANY (CANADA) LIMITED, GALT, ONTARIO

TODAY'S BIGGEST NEWS IN DEEP HOLE DRILLING


"O" RING SEAL
— prevents leakage out chuck.
SHANK
— eliminates water swivel.
COUPLINGS
— adequate coupling length.
RODS
— 2', 3', 4', 5', and 6' lengths.
BIT ADAPTER
— threaded for detachable bits.
BITS
— Timken tungsten carbide insert bits.
Gardner-Denver Automatic Feed Drifters—furnished with chromium plated air or water tube and three-way blowing valve for use with the "Ring Seal Shank."

GARDNER-DENVER DEEP HOLE DRILLING EQUIPMENT WITH "RING SEAL SHANK"

Here's a whole new set of field-proved drilling equipment that will revolutionize your deep hole drilling. Now, for the first time, you can take full advantage of tungsten carbide insert bits and modern rock drill power for faster, lower cost deep hole drilling.

No more water swivel troubles

The new Gardner-Denver "Ring Seal Shank" replaces the old type water swivel at a fraction of the cost—eliminates water swivel troubles.

Higher blowing pressures

"Ring Seal Shank" delivers high pressure water or air into the drill rods—assures positive hole cleaning.

Longer rod and coupling life

Gardner-Denver sectional rods and couplings withstand the shock and grind of deep hole drilling—last much longer than conventional equipment — because they're manufactured with the same skill and precision as rock drill parts. Rod design has been carefully engineered to minimize stress concentrations and to form a tight union between all parts.

Tested and proved in the field

Gardner-Denver deep hole drilling equipment has been widely field tested—is now in use in many United States and Canadian mines—drills prospect holes at a fraction of the cost of diamond drilling.

Write today for complete information.

GARDNER-DENVER

Since 1859

Gardner-Denver Company, Quincy, Illinois.

In Canada:
Gardner-Denver Company (Canada) Ltd.,
Toronto, Ontario.

It's Everyone's Business

APPLICATIONS for loan contracts for the exploration, development and mining of strategic and critical metals and minerals are now being accepted by the Department of the Interior. The RFC is authorized and directed by Executive Order 10161 to make such loans but the Secretary of the Interior must pass on the necessity for the loan. Applications should be addressed to James Boyd, acting deputy administrator, Defense Minerals Administration, Department of the Interior, Washington, D. C. rather than to the Field Offices of the Bureau of Mines as was first planned.

Application forms are not yet available but it has been learned that data accompanying the application should include the known quantity and grade of reserves, the anticipated rate of production, the equipment, supplies, and man power required for the additional production, etc.

Steel Capacity Expansion Controversy

A controversy has been in progress for the past few weeks between the industry and certain elements in Government over an increase in steel capacity. Secretary of Commerce Sawyer called in steel representatives in October and asked that they submit a program of steel expansion for the immediate future. The industrialists came back with a planned increase in steel-producing capacity of 9.4 million tons by the end of 1952.

Mr. Sawyer at first defended the industry plan for expansion against other prominent Government officials—among them Stuart Symington, chairman of the NSRB—who claimed that an increase in capacity of 20 to 30 million tons should be the goal. Mr. Sawyer, a former businessman, recognizes the realistic aspects of increased capacity in a period of only partial mobilization. While construction costs for new plants are probably the most important factor, the precarious manganese supply situation must be taken into account in increased steel production. Another factor is the possibility of a cut back in military demand at a time when civilian demand may be slackening. A steel capacity of 130 million tons would burden the industry with loan repayment and other costs on idle capacity in the event that demand dropped to 70 or 80 million tons annually.

Lately, Mr. Sawyer has softened and by the first week in November he was described as "not satisfied" with the first planned 10 pct increase in capacity.

It goes without saying that if present disturbances near the Manchurian border lead to open conflict with Chinese Communists, the advocates of a 30 pct increase in steel-producing capacity will carry the day.

Accelerated Amortization of Emergency Facilities

The National Security Resources Board has established a set of rules covering the issuance of Necessity Certificates pursuant to the acquisition of accelerated amortization rights for emergency facilities. Applications for a Necessity Certificate can be acquired from the NSRB, Washington 25, D. C., or from the Department of Commerce or any of its Field Offices.

Some of the criteria for determining the right to the complete 5-year amortization deduction include: (1) Whether the facility, or a part thereof, is "necessary in the interest of national defense during the emergency period." (2) Whether the facility is useful for other than defense purposes after five years. (3) Location of the facility with regard to military security.

Man Power

The mining industry has been acutely aware of the serious implications presented by the double impact of the military demand for men and the enticement of labor into higher-paying defense manufacturing industries.

On October 26 the Department of Defense issued its revised policies on the procedure for calling up Reservists in the Armed Forces. The new policies eliminate many inequities which formerly existed, such as the granting of only a week or two between the receipt of orders and the time for reporting for duty. From now on Reservists will receive at least four months warning on recall and will be allowed 30 days for reporting. This procedure will allow time for the proper consideration of deferment of key personnel engaged in critical occupations in the mining, milling, smelting, and refining industries.

Allocations of Essential Materials

On October 14 the National Production Authority issued its regulations covering the allocation of steel for defense purposes. Producers of certain steel products were given the option of rejecting a rated order (DO) received less than 45 days prior to the first day of the month in which shipment is requested. In addition, producers do not have to accept rated orders in excess of certain established percentages of their previous monthly shipments (from Jan. 1—Aug. 31, 1950) of certain designated steel products.

A supplement to the steel regulation order stated that directives will be issued to direct certain steel producers to accept orders for the production of steel designed for the construction and repair of freight cars. The supplement is supposed to assure the availability of steel for the freight car program.

Similar regulations covering the issuance of rated orders to the producers and fabricators of aluminum were issued on October 25. Aluminum producers were given a period of 60 days grace as compared to the 45 days given steel producers.

Regulations are expected soon which will cut the consumption of copper, zinc, aluminum, and nickel from 20 to 30 pct.

On November 1 the Government cut the export quota on copper for the 4th quarter of this year to 30 thousand short tons. This is a reduction of 20 thousand short tons from the earlier prevailing rate of export allowance. The serious short supply of copper led to the 4th quarter slash. Theoretically, since no export licenses for copper or copper products were issued in October, 15 thousand short tons of copper can be exported in November and December each.

The Department of the Interior announced on October 11 the appointment of Harold W. Swope as legal consultant in connection with minerals and metals activities of the Department under the Defense Production Act of 1950. Mr. Swope's duties will be to advise the Secretary on certification to the RFC of loans for the exploration, development, and mining of metals and minerals and on recommending to the General Services Administration purchase contracts for metals and minerals. Mr. Swope was chief counsel of the wartime Premium Price Plan for copper, lead, and zinc.

* George M. Humphrey, president of the M. A. Hanna Co. and of the Iron Ore Co. of Canada, has announced arrangements for financing the development of the Labrador-Quebec iron ore deposits in the total amount of $200 million. Plans call for immediately beginning the construction of mining, railroad, and handling facilities, and contemplate the start of ore shipments in 1955 which will reach 10 million tons a year shortly thereafter. Total drilled and proven ore now exceeds 400 million tons of high grade open-pit ore with prospects for large additional tonnages. Capital structure of the company authorizes up to $125 million of first mortgage bonds, $40 million of income debentures and $60 million of common stock.

Contracts for building the railroad have been let to Cartier Construction Co., Ltd., Montreal; Fred Mannix & Co., Ltd., Calgary; McNamara Construction Co., Ltd., Toronto; and Morrison-Knudsen Co., Boise, Idaho. The equipment is now being moved in and construction will begin before winter.

* COMINCO is starting an open pit at the Sullivan mine, Kimberley, B. C. Underground operations are continuing and will account for most of the production. Two million tons have been blocked out for open-pit mining.

* A Coal Defense Committee, organized by the bituminous coal industry, is set up to advise Congress and government agencies of manpower, machinery and supply, and transportation requirements needed to maintain an adequate supply of coal.

* Higher education is going to suffer if the three million men called for by the President to meet the 1951 military requirements are taken from the 19 to 26 age groups according to Raymond Walters, president of the University of Cincinnati.

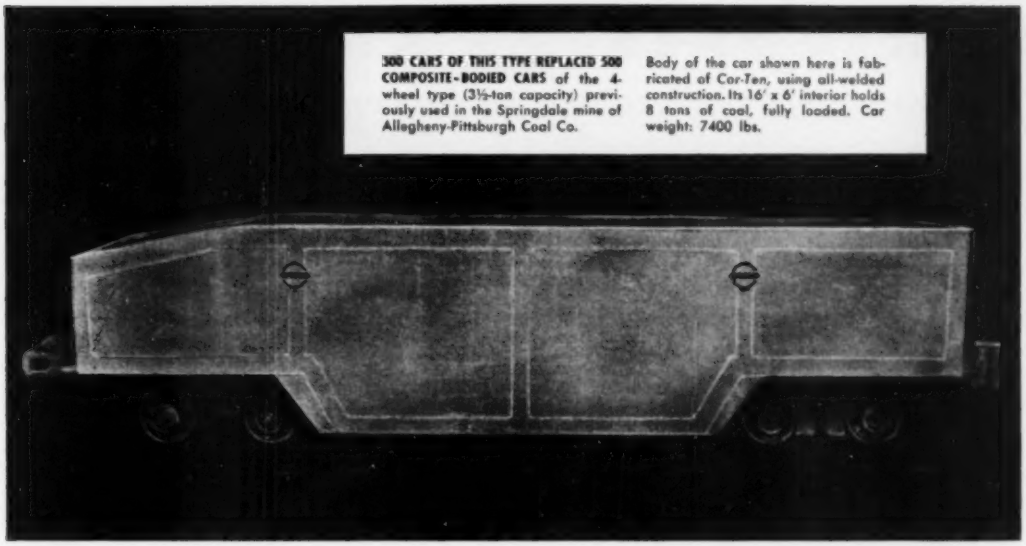
* Threatened shortages of first grade coking coal no longer presage diminishing supplies of high grade steel. James R. Killian, Jr., president of MIT, states that research sponsored by Republic Steel and the American Iron & Steel Institute has discovered processes for controlling brittleness in steel arising from high sulphur content.

* Average hourly output per man in the nation's mines has more than doubled between 1915 and 1950 according to the Bureau of Labor Statistics. Machinery and open-pit methods are accountable.

* Columbium, rare metal whose alloys are used in jet engine combustion chambers is in such short supply that the National Production Authority has stopped civilian use of the metal. Its principal source, Nigeria, is said to be nearly exhausted, and only lower-grade ores are being mined in the Belgian Congo and Brazil. Norway has formed a government mining company to exploit newly found deposits of soevite containing columbium.

* Iron ore shipments on the Great Lakes reached 72,979,597 tons on Nov. 6 according to the Lake Superior Iron Ore Assn. If the current rate can be maintained, shippers should reach their goal of 80 million tons before the shipping season is closed.

* Aluminum supplies for civilian use will be cut 35 pct on Jan. 1 to assure enough of the metal for military needs, according to William H. Harrison, administrator of the National Production Authority. He conceded that the slash would cause some unemployment.



500 CARS OF THIS TYPE REPLACED 500 COMPOSITE-BODIED CARS of the 4-wheel type (3½-ton capacity) previously used in the Springdale mine of Allegheny-Pittsburgh Coal Co.

Body of the car shown here is fabricated of Cor-Ten, using all-welded construction. Its 16' x 6' interior holds 8 tons of coal, fully loaded. Car weight: 7400 lbs.

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This eight-wheeled car of eight-ton capacity minimizes car changes at loading machines, permits greater operating flexibility and substantially cuts maintenance expenses...

Moreover, it combines superior corrosion resistance and increased strength through use of high strength, low alloy steel containing nickel, produced by Carnegie-Illinois Steel Corporation under the trade name *"Cor-Ten."*

All side plates, end plates, bottom plates and gussets in this modern mine car are fabricated of *Cor-Ten* by the manufacturer, Irwin Foundry & Mine Car Company of Irwin, Pa.

During recent years, the use of this type of steel has increased at a remarkable rate. Produced under various trade names by leading steel companies, high strength steels containing nickel along with other alloying elements, provide three basic advantages for mine car construction:

1. **High strength, permitting increased strength (where section thickness remains the same as that of the carbon steel it replaces) or, in many cases, weight reduction.**
2. **Excellent behavior in fabrication, readily forming and welding.**
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EMBLEM OF INCO SERVICE

Because of unusually heavy industrial and defense demand, rationing of nickel has been in force since July 1st. However, we believe that dissemination of technical data and service experience can help to promote the intelligent utilization of critical materials, so essential in these times. We shall, therefore, continue to issue information on new developments and user experience with nickel-containing materials.

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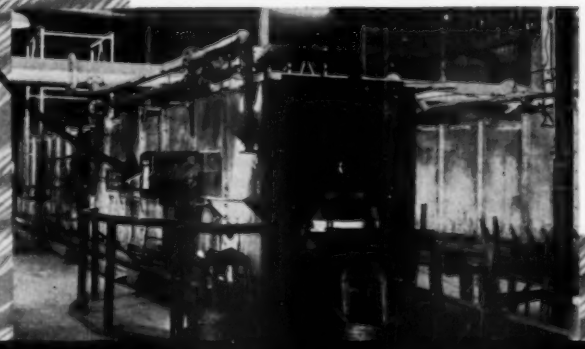

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Birmingham – A City With a Future

COURTING new industries is the time honored duty of the chambers of commerce; but Birmingham, until recently, has done its courting without the benefit of troubadors. This is all changed now for the iron and steel industry is singing an enticing song about steel priorities for new industries from Birmingham's own mills. Along with the increasing industrialization of Birmingham there has been a marked influx of industry into the Southeast and neighboring areas in the south central states which augurs well for business in Birmingham. This important stronghold of the foundry industry (consuming 95 pct of Birmingham's merchant pig iron in 1950) has been the source of important announcements about additions to steel ingot capacity. The virtual abolition of basing point pricing gives the Birmingham iron and steel industry an advantage in the South's new markets. This means more business for Birmingham, but the problem of raw materials supplies is posed.

Steel company planners are moving in the stratosphere of high financial and international negotiation; but on the sweaty production level engineers are working out the technical details. A glimpse into the stratosphere shows U. S. Steel and Bethlehem Steel planning large scale mining of iron ore in Venezuela; some of this ore will find its way to Birmingham furnaces. There are rumblings in Birmingham about furnaces in Mobile; but strategic considerations affecting Venezuelan ore supply are a serious handicap. Republic has been concentrating ore at Spalding for some time, but the future of this operation may be tied up with Republic's negotiations for Venezuelan ore.

Mining Engineering this month tells the story of what mining engineers are doing to get more iron ore and coal out of the ground cheaper and more safely. It is a story about mechanization; careful planning and development of mines to adopt mining methods to the use of machines; and low cost materials handling methods. Birmingham coal and ore mining engineers have worked together and have been quick to adapt each others methods where advantageous. First in the nation in the percentage of coal mechanically cleaned in 1948, Birmingham coal operators are improving and adding washing facilities. The problem of ore concentration and agglomeration is being studied, and the future may see more companies concentrating Red Mountain ore.

The work of the mining engineers and their associates who have written this issue is closely related to the future of Birmingham, and in a limited but extremely important way they will shape this destiny. In the following pages is the saga of what has been accomplished and it is indicative of what is to come.

The Economics of Raw Material Supplies in the Birmingham District

by
E. C. WRIGHT
Head, Dept. of Metallurgical Engineering University of Alabama

FOR many years the cost of making pig iron and steel in the Birmingham district has been about the lowest in the United States. The close proximity of the important raw materials such as coal, iron ore, limestone, and the large available scrap supply made it possible to assemble all of the important ingredients necessary for steelmaking with only nominal costs for transportation. Both iron ore and coal were being mined in this district at one time for about $1 per ton from underground operations. At that time mining wages were low everywhere and the labor rate in the South was considerably below that of other mining districts. This situation persisted until the emergence of labor unions as an economic and political force, emigation of southern labor to northern plants, and continued growth of industry in the South in the years following the depression of 1932. Since that time union organizing of miners and steel workers throughout the nation has increased labor costs in mining and steelmaking. This influence has been felt particularly in underground mining operations since the proportion of labor cost to total cost is still high. The previous advantage in basic labor rate in the South has been almost entirely eliminated by the equalization of labor rates in basic industries by national unions.

Since 1945 these influences have been especially noticeable in increasing the cost of mining coal and iron ore in the South. The northern steel districts are still being largely supplied with iron ore from the Minnesota open-pit mines and the cost of this ore has not increased nearly as much as the cost of ore obtained from underground mines in Birmingham. A similar situation exists in coal mining; the thinner seams in Alabama are not so amenable to mechanization as the northern mines and, as a result, the output per man-day in southern mines is only about three tons of coal as compared to five or six tons per man-day in the large captive mines of West Virginia and Pennsylvania which serve the northern steel plants. These factors have raised the cost of making pig iron and steel in Alabama to levels that are coming close to the costs experienced in the older northern steel mills.

Two factors still give the Birmingham district a favorable position with respect to the other steel districts. These are the relation of the adjacent market and the increased cost of transporting raw materials to the steelmaking centers. In the last thirty years the increasing tempo of industrialization in the southern states has greatly augmented the existing market for steel products. Tremendous expansion in the oil and natural gas industries in the southwestern states has created a large market for steel which is still largely supplied by northern plants. This is particularly noted in respect to sheet steel and pipe. At the present time it is estimated that there is a market for steel which could be serviced by southern steel mills that is almost three times the present output of steel in the southern mills.

Northern steel mills are absorbing increased costs of transportation of both iron ore and coal to producing centers and increased freight costs on finished steel to new consuming steel points in the South. Chicago has to face increased costs for coal shipping from Kentucky and West Virginia mines; Pittsburgh has large increases in cost of shipping iron ore from Minnesota; and intermediate plants like those in Buffalo, Youngstown, Lorain, and Cleveland have encountered increases in transportation of both iron ore and coal. Both rail and lake transport have increased each year since the end of the war and these factors have had a profound effect on the cost of producing steel in the old established steel centers. Since these northern plants have also had increases in labor costs, the combination of these two factors has had a compensating effect on the developments which have increased the cost of making steel in the Birmingham district.

In analyzing the cost of assembling raw materials for making steel it is necessary to study the cost of bringing each important ingredient to the steel plant. The cost of mining each constituent

Iron ore tonnages are in long tons, 2240 lb, and other tonnages are short tons, 2000 lb.

Analyzing the cost of raw material supplies for iron and steelmaking in the Birmingham district requires discussion of trends in procurement and cost for the major steel producing districts. The expert analysis presented here is the result of a 3-year survey conducted by the author for the Bureau of Business Research of the University of Alabama.

may vary widely in a given field and it becomes necessary to estimate the cost of production of a number of producers and then take a weighted average of the whole district. For example, Lake Superior iron ores are obtained from open-pit mines to the extent of about 85 pct of the total production and 15 pct from deep underground mines. Some of these ores are then subjected to ore dressing operations to improve their quality to that of blast furnace grade. At the present time it is reported that nearly 25 pct of all the Lake ores are being treated by one or more beneficiating processes. A weighted average cost of the Lake ore now being consumed in northern steel plants must include these factors. To this figure is then added the cost of transporting the ore by ship and rail to the steel plant. Only a rough approximation can be thus obtained since the relative cost of underground and open-pit mining in the many different mines can only be guessed at and the cost of beneficiation methods varies.

Coal and Coke

A similar situation exists in respect to the cost of coal and coke. The average cost in each district represents a weighted average of all the reporting mines and includes data from strip mines, small wagon mines, and the highly mechanized mines of the larger coal producers.

The cost of blast furnace coke is complicated, depending not only on the cost of the coal charged but also on the value of the by-products of the coke oven. Two of the most important by-products, coke oven gas and tar, are usually consumed as fuels in the steel mills, and their credit value is often calculated on a Btu basis in comparison with the cost of coal at the point of use. At points such as Baltimore and Chicago, where the cost of coal, shipped for long distances, is high, the by-product credit is high; at Birmingham and Pittsburgh, where coal is much cheaper, and in Texas, where natural gas and fuel oil is cheap, the by-product credit for these two items is much lower. This difficult accounting procedure often gives an odd result for the net cost of blast furnace coke. A study of the steel industry in the southern states was started in 1947 by the Bureau of Business Research of the School of Commerce and Business Administration of the University of Alabama. Difficulty was encountered because the period 1947 to 1950 was one of chaotic costs. An example of the wide fluctuations in material costs is exhibited in iron and steel scrap prices; this commodity varied between $20 and $46 per ton during 1949 and a similar spread in the price of scrap occurred in the spring of 1950. Freight rates on all raw materials were raised several times and the cost of mining ore and coal increased each year as the union contracts were renewed. Strikes, work stoppages, and shortages of many materials have also contributed to increased costs. During this three year period many discussions have been held with steel producers in the South and East; railway, barge, and ocean shipping operators; and trade associations and government bureaus such as the American Iron & Steel Institute, National Coal Assn., State Mining Institutes, and the U.S. Bureau of Mines.

Early in this survey it was realized that the effect of the greatly increased cost of mining low grade iron ore in the Birmingham district and the rapid depletion of the high grade open-pit ores of the Lake Superior region would have a critical influence on the cost of making steel in both the northern and southern steel districts within the next few years. Consideration of these two problems led to a review of the developments in the concentration of low grade ores in both districts and also to a study of the intensive activities of the major steel companies in the exploration of foreign ore deposits.

A concise summary of the history, development, and possible application of beneficiation of southern ores has recently been made by E. H. Rose, research engineer, Tennessee Coal, Iron & Railroad Co. (Preprint, Regional Technical Meeting, October 1949, American Iron & Steel Institute, pp. 221-242.) This author gives an accurate description of the physical features of Red Moun-

tain ore which have prevented economic concentration. At least 25 years of research have gone into this effort and all but one of the concentrating processes tried have been abandoned; the Spaulding mill of Republic Steel Corp. is still operating but the cost of this process has not definitely proven the success of the undertaking.

The reason for this lack of success lies in the distribution of the iron and gangue bearing minerals in the Red Mountain ore. Mr. Rose reports that "the iron mineral in both ores is a soft earthy hematite, which tends to become 'paint' upon the slightest provocation, especially when wet. In the sandstone, the individual silica grains are imbedded in the continuous matrix of noncrystalline, clay-containing hematite." A separation by grinding requires a rubbing or scuffing action to rub off the iron bearing cement from the silica grains and no grinding mill is available to do this efficiently. Moreover, the iron mineral so separated is usually smaller than 150 mesh. If the grinding is done dry a difficult dust collecting problem arises, while if wet grinding is employed, most of the iron mineral is obtained as a slime which imposes a tremendous dewatering problem for any plant handling thousands of tons of ore per day. A production of 6 million tons of *red ore* per year will require concentrating plants capable of producing 25,000 tons of concentrate per day.

Mr. Rose speculates in his paper regarding the feasibility and economics of subjecting crushed red ores to a magnetic roast and then separating the magnetized iron mineral from the nonmagnetic gangue mineral by passing the treated ores over magnetic separators. This is the most efficient and cheapest method of concentrating iron ores and is used with great success wherever the iron occurs in nature as the magnetic oxide Fe_3O_4. A whole industry has been built on this process in New York state and some 3 million tons of sinter containing over 60 pct iron are now being produced from mines idle since colonial days.

Magnetic Concentration

The adaptation of magnetic concentration to the Birmingham ores will involve several steps: (1) fine grinding (dry), (2) magnetic roast at 1100°F in reducing gas to convert hematite to magnetite, (3) wet grinding and magnetic separation of magnetite from gangue, and (4) sintering or some agglomeration method of making a suitable blast furnace feed of the concentrate.

Both the magnetic roast and the sintering operation require fuel and special furnace equipment. Laboratory studies by the Bureau of Mines and several mining companies have shown that a concentrate containing up to 55 pct iron can be made by this process with an iron recovery of 92 pct of the iron in the low grade ore charged. If such a material can be substituted for the 35 pct iron ore now being smelted in Birmingham, Mr. Rose estimates an increase in blast furnace output of 48 pct, a decrease in coke consumption of 849 lb per ton of iron produced, a decrease of fluxing stone of 714 lb per ton, a decrease in slag volume of 1700 lb per ton of iron. This improvement in smelting practice is estimated to save nearly $6 per ton of iron or about $3 per ton of 55 pct iron concentrate. These large savings should go a long way towards paying for the cost of magnetic concentration of Birmingham ores if the engineering and operating difficulties may be overcome.

A pilot plant test of this practice was made several years ago at the Red Mountain Ruffner mine by Sloss-Sheffield Iron & Steel Co. Although a successful magnetic roast was conducted and a concentrate of satisfactory grade was produced, the output of the plant never reached expectations and difficulty in the operation of the magnetic separators resulted in high costs. This outcome led to the abandonment of the project but interest in this process is still high in the Birmingham district.

In the Lake Superior districts, the beneficiation of low grade iron ores is also a vital question in relation to the ore mining industry and the survival of the northern steel industry. The reserves of direct shipping ores which can be mined in open-pits have been rapidly depleted in the past ten years and, if production at present rates continues, most of this cheap ore will be consumed in another ten years according to existing estimates for reserves. At the present time ore from open-pit mines costs about $2.75 per ton at the mine, whereas ore of equivalent quality costs over $4 per ton when obtained from underground mines. Practically all of the Wisconsin and Michigan ore now comes from underground mines and increasing amounts will have to be mined by this method in Minnesota unless new deposits of open-pit ore are developed. In 1948 about 85 pct of the Lake Superior ore was obtained from open-pit mines. If 50 pct of the 1948 ore production had been obtained from underground mines the increase in cost of all the ore used would have been about 45¢ per ton of ore which would represent about $1 per ton in the cost of pig iron.

Geologists have repeatedly estimated that over 70 billion tons of taconite ore containing 25 to 40 pct iron is available in the Lake Superior region. The chemical composition of the taconite ores is similar to that of the ferruginous sandstones of the Birmingham district but the physical character of the taconites is radically different. The taconite ore is much harder, making it more difficult to mine and crush as compared to the softer ores of the Red Mountain deposits. The iron bearing minerals are also in the form of separate particles and not as amorphous mud or cement which is so typical of the oolitic ores in the South. This feature results in a better liberation of the iron minerals from the silica in grinding and makes this material more amenable to ordinary ore dressing practices.

Here again the main factor of successful ore dressing of taconite entails fine grinding to get a good mechanical separation of the hematite mineral from the silica. The fine grinding of the hard ore is costly and also gives rise to slime conditions which make an unsatisfactory situation for gravity separation of the minerals. As a result the greatest interest at the present time seems to be in the flotation or magnetic separation methods for processing taconite ores, since these two processes are adapted to the separation of the minerals in finely ground pulps. Thorough studies have been conducted by the U.S. Bureau of Mines, Minnesota State Mines Experiment Station, and several ore mining companies of these two processes both in the laboratory and on a pilot plant basis. The cost of the concentrates

produced has always been well above that of direct shipping ores and, as long as these are available, no large scale installation of taconite concentrating plants is economically feasible.

It has been estimated that an investment cost of at least $20 per annual ton of concentrate will be required even for large scale operations. Amortization cost alone would amount to $1 per ton. If most of the 90 million tons of 50 pct iron ore now being shipped from the Lake Superior region annually were to be replaced with concentrated ore, the total investment needed would approach $2 billion. The added cost of grinding, magnetic roasting, and sintering would amount to at least $2 per ton of concentrate. If the iron content of the taconite concentrate is 50 pct, the northern steel industry will be faced with an increased cost of $3 per ton for a 50 pct iron concentrate. From this there will result little saving in shipping costs or smelting cost. A concentrate containing 60 pct iron, however, would cause such a large saving in shipping and smelting costs as to partially compensate for the cost of concentration. It will be evident that the efficiency of the concentrating process will largely determine the outcome of this important problem. The northern steel plants are therefore obviously faced with higher impending iron ore costs either from more underground mining or beneficiation.

In the southern region the outlook is not nearly as gloomy. The district has long operated on ores from underground mines. It has also produced cheap pig iron from 35 pct iron ores. The introduction of successful beneficiation processes to raise the iron content of the concentrate to 50 pct will result in such a large savings in smelting costs at the blast furnace that there should be little increase in the over-all cost of pig iron in the district.

Foreign Ore Deposits

The difficulties associated with the beneficiation of both Lake Superior and southern iron ores has recently led to greatly increased interest in the development of foreign ore deposits. An economic study of this phase of the iron ore problem introduces a great difficulty in any cost analysis because of the erratic situation in ocean shipping costs. This is particularly true of vessels sailing under the American flag as the high labor standards established for American seamen, as compared to those prevailing on foreign lines, make the daily operating costs of American vessels much higher than those of foreign competitors. Shipping companies estimate that it costs between $1500 and $2000 per day to operate a 10,000 ton freighter. Moreover, the tremendous tonnage of iron ore which must be moved (from 2 to 10 million tons annually) makes it almost necessary for the ore shipper to organize his own shipping line. This factor will almost certainly require a tremendous investment cost in organizing the shipping needed for the movement of large tonnages of foreign ores and an amortization figure of between $1 and $3 per ton of ore will have to be faced in supplying the mining plant, port facilities, and ships needed for foreign ore development; such a figure has been included in the mining costs estimated for the foreign districts.

The most interesting foreign iron ore deposits for American steel producers are those in Canada, Chile, Brazil, Venezuela, Africa, and Mexico. Although there are also large deposits in Cuba, repeated exploration and use of the Cuban ores has continually resulted in the projects being abandoned during periods of close competition. There are large deposits of high grade iron ores in the countries given above and the extent and characteristics of these deposits have been described by T. W. Lippert, "Cerro Bolivar," *Mining Engineering*, February 1950.

The enormous deposits of high grade iron ore recently discovered in Labrador (J. A. Retty, "North America's Greatest Iron Ore Field," *Mining & Metallurgy*, September 1948), which are now undergoing systematic exploration, have attracted great interest. At the end of the summer of 1948 over 300 million tons of ore containing over 55 pct iron have been proven and the deposits are now estimated to contain over 2 billion tons of reserves. This is twice the total estimated reserves of the Lake Superior region. Several mining and steel producing companies have organized to exploit these ores. Delivery to existing steel producing centers will involve mining under difficult climatic conditions, the construction of a railroad 300 miles long to the St. Lawrence River, and the shipment of the ores from a river port by rail or a combination of rail and boat to steel plants. A large investment in mining equipment, transportation, and storage depots will be involved in this development, but present estimates indicate that these ores may be delivered to steel plants at a cost only slightly higher than the present delivery cost of Lake Superior ores.

The large and rich ore bodies of Brazil seem to be so unfavorably situated geographically that their use in American steelmaking does not seem at all feasible at present. These ores are located about 300 miles from the nearest Brazilian ocean port, must be shipped on a narrow-gage railroad at high cost and are then at least 4500 miles from the nearest American steel center. It is estimated that the cost of shipping from the mine to the blast furnace is betwen $9 and $12 per ton, a prohibitive figure at this time. In 1950 Brazilian ore of 67 pct iron content is quoted at 24¢ per iron unit or $16.08 per ton. The cost delivered to Birmingham must be at least 21¢ per unit, and, since the mining cost should not exceed $1.50 per ton, the effect of high transportation cost is obvious. These shipping costs may be greatly reduced by huge ore boats carrying 20,000 to 40,000 tons.

The Chilean ores have been used for many years by the Bethlehem Steel Co. at its Sparrows Point plant near Baltimore; before World War II and since the close of hostilities, about two-thirds of the ore used at this plant originated in Chile. Although the distance is nearly 9000 miles, the ore deposits are near the coast in Chile and are unloaded directly from the ship to the blast furnace stockpile at the plant. Ocean shipping costs were reduced by the use of large ore carriers holding over 20,000 tons and this enabled the company to deliver this high grade ore (62 pct iron) at quite favorable costs. Most of the original deposit is reported to be exhausted, but the company is now developing a similar ore body in Chile which is stated to contain about 20 million tons. In 1948, over 2,600,000 tons of Chilean ore were imported.

Table II

Assembly Cost of Raw Materials Per Ton of Pig Iron Produced at Various Locations

July 1, 1950

Location	Ore cost at mine per net ton[a]	Freight mine to furnace	Ore cost per unit iron[c]	Tons ore per ton pig iron	Ore cost	Flux cost	Ore and flux cost	Coal cost at mine per ton	Freight mine to furnace	Tons coal per ton of coke	Tons coal and coke per ton of pig iron	Coal cost	By-product credit on cost of coke	Net cost of coke	Total assembly cost	Overhead cost above materials	Total[g] cost pig iron
Youngstown, Ohio Lake Ore Pittsburgh Coal	$2.62[b]	$3.88	$0.144	2.00[e]	$13.00	$0.80 0.4 tons	$13.80	$5.00	$1.85 Barge and Rail	1.55	Coal 1.40 Coke 0.90	$9.59	$2.30	$7.29	$21.09	$3.00	$24.09
Pittsburgh, Pa. Lake Ore Pittsburgh Coal	2.62[b]	4.28	0.153	2.00[e]	13.80	0.80 0.4 tons	14.60	5.00	0.40 Barge	1.55	Coal 1.40 Coke 0.90	7.56	2.16	5.40	20.00	3.00	23.00
Pittsburgh, Pa. 80 pct Lake Ore 20 pct New York Ore Pittsburgh Coal	2.62[b] 5.32	4.28 3.13	0.153 0.141	1.85	13.34	0.70 0.35 tons	14.04	5.00	0.40 Barge	1.55	Coal 1.32 Coke 0.85	7.13	2.06	5.08	19.12	3.00	22.12
Baltimore, Md. 20 pct Lake Ore 30 pct Swedish 50 Chilean Ore	2.62[b] 2.00[d]	5.12 5.35	0.172 0.140 0.132	1.73	2.58 3.65 6.40 12.63	0.60 0.30 tons	13.23	5.11	4.63 Rail	1.40	Coal 1.12 Coke 0.80	10.91	1.21	9.70	22.93	3.00	25.93
Lorain, Ohio Lake Ore Ky. & W. Va. Coal	2.62[b]	2.70	0.117	2.00[e]	10.64	0.52 0.35 tons	11.16	5.11	3.54 Rail	1.40	Coal 1.26 Coke 0.90	10.90	1.74	9.56	20.72	3.00	23.72
Chicago, Ill. Lake Ore Ky. & W. Va. Coal	2.62[b]	2.70	0.117	2.00[e]	10.64	0.52 0.35 tons	11.16	5.11	4.24 Rail	1.40	Coal 1.26 Coke 0.90	11.78	1.48	10.30	21.46	3.00	24.46
Birmingham, Ala. 100 pct Red Ore Alabama Coal	3.75	0.35	0.124	2.70	11.07	0.30 0.20 tons	11.37	6.00[f]	0.40	1.54	Coal 1.85 Coke 1.20	11.85	2.45	9.40	20.77	3.25	24.02
Birmingham, Ala. 50 pct Ala. Red Ore 50 pct Venezuelan Ore Alabama Coal	3.75 2.50	0.35 5.95	0.124 0.128	2.10	13.19	0.30 0.25 tons	13.49	6.00[f]	0.40	1.54	Coal 1.39 Coke 0.90	8.90	1.80	7.10	20.59	3.00	23.59
Mobile, Ala. 50 pct Red Ore 50 pct Venezuelan Ore Alabama Coal	3.75 2.50[d]	2.00 4.17	0.184 0.115	2.10	13.04	0.30 0.25 tons	13.34	6.00[f]	1.40	1.54	Coal 1.39 Coke 0.90	10.28	2.03	8.25	21.59	3.00	24.59
Texas Ore Oklahoma Coal East Texas	5.50[d]	0.60	0.137	2.00	12.20	1.50 0.70 tons	13.70	6.00	1.25	1.56	Coal 1.40 Coke 0.90	10.15	0.83	9.32	23.02	3.30	26.32

[a] *Average ore cost for underground and open-pit mines per short ton.*

[b] *Includes cost of mining and amortization of equipment and transport to seaboard.*

[c] *Includes prorated estimate of mining, washing, calcining, and sintering per ton of 50 pct iron concentrate.*

[d] *Value of ore per long ton at assembly point divided by the percentage of iron in the ore.*

[e] *Includes mining and washing cost.*

[f] *Average analysis of Lake Superior ores shipped 50.3 pct.*

[g] *Does not include credits for blast furnace by-products, such as gas, slag, flue dust.*

[h] *Iron ore tonnages are converted from long tons to short tons for computation purposes.*

Table IV

Cost of Basic Open Hearth Ingots Per Ton, July 1950[1]

	Cost 1 Ton Pig Iron	Pig Iron Used[2]	Home Scrap[3]	Purchased Scrap[4]	Iron Ore	Flux	Additions[5]	Total Materials	Fuel,[6] Labor, etc.	Total Cost
Youngstown, Ohio	$24.09	$13.40	$8.42	$8.75	$0.58	$0.35	$1.65	$33.15	$6.70	$39.85
Pittsburgh, Pa.	23.00	12.80	8.12	8.75	0.61	0.35	1.65	32.28	6.70	38.98
Baltimore, Md.	25.93	14.56	8.93	7.50	0.62	0.35	1.65	33.61	6.70	40.31
Lorain, Ohio	23.72	13.35	8.33	7.50	0.50	0.35	1.65	31.68	6.70	38.38
Chicago, Ill.	24.46	13.75	8.55	7.50	0.50	0.35	1.65	32.30	6.70	39.00
Birmingham, Ala.	24.02	13.47	8.40	6.50	0.40	0.35	1.65	30.17	6.70	36.81
Mobile, Ala.	24.59	13.66	8.75	6.50	0.55	0.35	1.65	31.46	6.70	38.16
East Texas	26.32	14.76	9.06	6.25	0.55	0.35	1.65	32.62	6.70	39.32

[1]*Based on 50 pct scrap charge and 90 pct yield of metallics charged.*
[2]*Based on 50 pct pig iron @ 90 pct yield of metallics charged.*
[3]*Based on 25 pct home scrap @ 90 pct yield. Valued at cost of pig iron + $6 per net ton conversion cost.*
[4]*Based on 25 pct purchased scrap @ 90 pct yield. Valued at average Iron Age scrap price, Jan. 1 - July 1, 1950.*
[5]*Fuel costs fluctuate widely in different districts and are too involved for accurate estimates.*
[6]*Based on average additions of ferrosilicon, ferromanganese, and aluminum per ton of ingots.*

Birmingham's raw material cost per ton pig iron is bettered by Pittsburgh and southern Ohio plants but with blast furnace charges using Venezuelan ore and red ore, 50-50, it is competitive.

Data for these tables on the cost of raw material assembly for the major steel producing districts was obtained from personal interviews with steel producers in the South and East; railway, barge, and ocean shipping operators; and trade associations and government bureaus such as the American Iron & Steel Institute, National Coal Ass., State Mine Institutes, and the U. S. Bureau of Mines. Cost figures are given in dollars per net ton (2000 lb).

Table I

Special Coal and Coke Cost Per Ton, July 1950

	Cost coal per ton at mine	Freight per ton to furnace	Tons coal per ton of coke	Cost of Coal per ton of coke	Cost above coke ovens ton of coke	By-product Credits Per Ton Coke: Gas[2] cu ft		Tar[3] gal		Ammonium Sulphate[4] lb		Light oil[5] gal		Coke breeze	Total credit by-products	Net cost coke per ton
Youngstown, Ohio	$5.00	$1.85 Barge & Rail	1.55	$10.62	$2.15	11,500	$1.37	15.0	$1.50	36	$0.76	5.5	$0.88	$0.20	$4.71	$8.06
Pittsburgh Coal																
Pittsburgh, Pa.	5.00	0.40 Barge	1.55	8.37	2.15	11,500	1.20	15.0	1.50	36	0.76	5.5	0.88	0.20	4.54	5.98
Pittsburgh Coal																
Baltimore, Md.	5.11	4.63	1.40	12.64	2.00	9500	1.78	6.5	0.65	20	0.42	3.0	0.48	0.20	3.53	12.11
West Va. Coal																
Chicago, Ill.	5.11	4.24	1.40	13.09	2.00	9500	1.71	7.5	0.75	22	0.46	3.3	0.53	0.20	3.65	11.44
E. Ky. & W. Va. Coal																
Lorain, Ohio	5.06	3.54	1.40	12.56	2.05	10,000	1.66	9.5	0.95	25	0.53	4.0	0.64	0.20	3.98	10.63
40 pct W. V. }																
60 pct Pennsylvania }																
Birmingham, Ala.	6.00[1]	0.40	1.54	9.86	2.15	10,900	1.44	12.0	1.20	30	0.63	4.5	0.72	0.20	4.19	7.81
Alabama Coal																
Mobile, Ala.	6.00[1]	1.40	1.54	11.42	2.15	10,900	1.66	12.0	1.20	30	0.63	4.5	0.72	0 20	4.41	9.16
Alabama Coal																
East Texas	6.00	1.25	1.56	11.30	2.15	11,800	0.60	16.0	0.80	28	0.59	5.6	0.89	0.20	3.08	10.37
Oklahoma Coal																

[1]*Includes mining and washing costs. Estimated average from National Coal Assn. and Bureau of Mines figures.*
[2]*Gas credit based on 500 Btu gas at value of 1 million Btu at point of use except in E. Texas where coke oven gas competes with natural gas.*
[3]*Coal tar @ 10¢ per gal. except in E. Texas where tar is credited @ 5¢ per gal.*
[4]*Ammonium Sulphate @ 2.1¢ per lb.*
[5]*Light oil derivatives @ average value of 16¢ per gal.*

As has been pointed out previously, there is at present great interest in the ore deposits in the Orinoco River region of Venezuela; two of the largest steel companies are now exploring and developing them. The mines are some 300 to 400 miles from the mouth of the Orinoco River and are only about 3000 miles from such ports as Mobile or Baltimore and Philadelphia. The study of the raw material cost assembly of these ores at Mobile and Birmingham is therefore included, although these data are entirely speculative since none of the ores have been actually shipped up to this time. Fairly reliable figures are available for the costs of Swedish, Brazilian, and Chilean ores as considerable tonnages have been consumed in this country. Mr. Lippert's article, previously cited, gives much new information about the status of the Venezuelan ore deposits. The extensive programs projected by the two largest steel producers clearly indicate that ore may be delivered to existing steel plants at a favorable cost. The estimated cost of these rich ores at Birmingham and Mobile, given in Table II, confirms this assumption.

An interesting speculation as to the origin of iron ores used in the Great Lakes, Pittsburgh, Youngstown and Ohio Valley steel centers in 1960 is presented by Henning and Braund in the *American Iron & Steel Institute Yearbook 1950*.

The authors estimate that ten years from now these northern plants will be using 10 million tons of taconite concentrate and 20 million tons of imported ores, mostly from Labrador and Venezuela. If the projected increase in foreign ores now prophesied for the Alabama district is added to this, it will be evident that between 30 to 40 pct of the total iron ore will come either from beneficiation plants or from imports. This prophecy is emphasized by the great increase in ore imports since 1945. A recent report states that 8,300,000 tons were imported in 1949.

As a result of this survey a fairly logical estimate of the cost of assembling the basic raw materials such as coal, coke, fluxing stone, iron ore, and scrap was obtained for each of the important steelmaking centers of the southeast and northern districts. These estimates were revised each year as the increases in labor and shipping costs appeared and the figures were finally frozen as of July 1, 1950. By the time of this publication further increases will have probably occurred but it is believed that the tabulated results represent a comparative view of the present economic position of the more important steel producing centers. The cost estimates are also based entirely on a fully integrated operation which permits the most efficient use of by-products such as blast furnace gas, slag, and flue dust, coke oven chemicals and fuels, etc., and could not be achieved in any small steel plant making less than 2 million tons of ingots per year. The data compiled are exhibited in Tables I, II, III, and IV.

It is significant that the estimated cost of producing pig iron and steel in the important eastern and southern steelmaking districts do not differ greatly and the variations that do occur may well be due to differences in the ability of management in various plants or in the efficient use and disposal of by-products. The cost of certain raw materials such as coal and coke at the different points shows a wide variation and this is mainly due to transportation costs. Table I emphasizes this factor and indicates the unfavorable position of such points as Chicago and Baltimore.

A few of the outstanding features with reference to the relative position of Birmingham among the major steel centers may be mentioned. Of the eight districts studied in this survey, as shown in Table II, east Texas has the highest total assembly cost, $23.02 per ton of pig iron produced. This compares with Birmingham, $20.77; Lorain, $20.72; Pittsburgh, $20.00 or $19.12, depending on ore used; Chicago, $21.46, and Baltimore, $22.93. The plants nearest to the coal fields have the lowest costs and this situation seems destined to prevail.

As an illustration of the tremendous increases of the cost of both raw material and finished steel during the past five years, Table III was prepared to show the difference between 1945 and 1950.

Table III

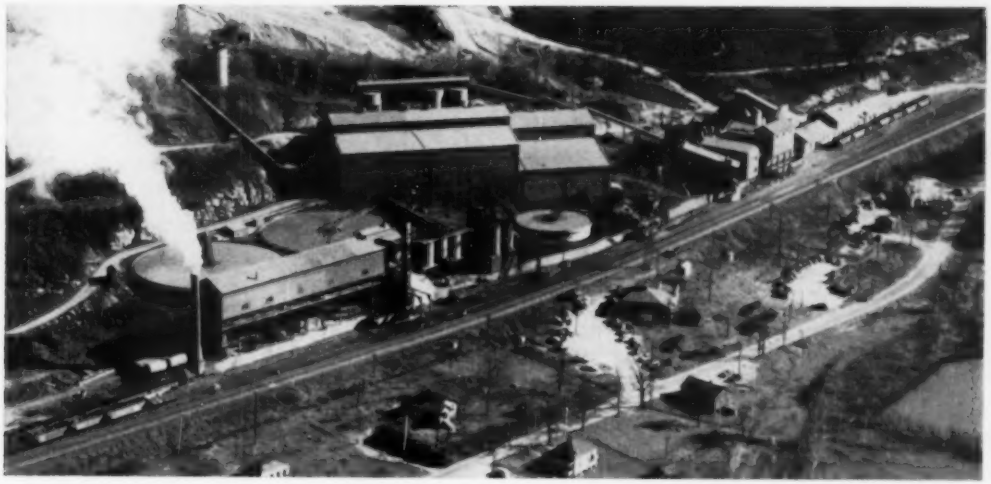
Comparative Cost Ratios Between 1950 and 1945

	Iron Ore	Coke	Pig Iron	Steel
Pittsburgh	2.10	1.32	1.73	1.76
Baltimore	2.18	1.40	1.94	1.84
Lorain	2.04	1.43	1.87	1.76
Chicago	2.04	1.44	1.93	1.84
Birmingham	1.73	1.58	1.68	1.79

Table IV exhibits the estimated cost of making steel ingots. This is based on a 50 pct pig iron and 50 pct scrap charge since this mixture represents the average charge used in American open hearth plants. The pig iron cost is taken from Table II and the purchased scrap charge is based on the average price of scrap for the first six months of 1950 for each consuming center. The scrap used in the charge is half home scrap and half from outside sources. Since both pig iron and steel scrap are cheaper in Birmingham than elsewhere the lowest steelmaking cost appears in the southern plants. For many years southern scrap has been shipped in large tonnages to northern steel mills in spite of the high freight cost involved in such long hauls. This situation indicates a lower price for scrap at Birmingham and a surplus supply in the South.

There has been considerable discussion lately as to the desirability of Mobile as a steelmaking location. Such discussions must hinge to a considerable extent on the availability and cost of foreign ores in respect to domestic. To ship Alabama ore and coal to Mobile would obviously be much more costly than smelting at Birmingham. The estimates appearing in Tables II and IV for Mobile are based on an ore blend of 50 pct Venezuelan and Alabama ore. The use of this rich ore would so greatly reduce the coke and smelting costs that a favorable figure is obtained. These estimates are entirely speculative as no Venezuelan ore has been shipped and the delivered cost of this ore at present is uncertain. It is not believed that the use of a 100 pct burden of Venezuelan ore on the blast furnace would be desirable as the slag volume produced would be so small as to give poor control over the sulphur content of the pig iron. However, as underground mining cost of Birmingham red ore continues to increase, the feasibility of the foreign ores at Mobile becomes greater and this port is also an attractive point for the shipment of finished steel to export markets and coastal consuming centers.

Bureau of Mines Studies Iron Ore Concentration—Gravity-Flotation Combination Appears Best



Republic Steel Corp. iron ore concentrator at Spalding is the only one operating at present in the Birmingham district.

by **Ballard H. Clemmons** acting chief, Metallurgical Br., Minerals Technology Div., Region VII, Bureau of Mines, U. S. Dept. of the Interior

THE future of the steelmaking industry of the Birmingham, Ala., district is closely related to and, in a large measure, dependent on the development of workable, economic processes of ore concentration. Operations thus far in the area have been possible only because of the fortuitous occurrence of iron ore, fluxing stone, and coking coal—all the primary iron-making materials—within a 25-mile radius of the blast furnaces.

Commercial ore of the district consists mainly of a 10 to 20-ft hematite-calcite-silica bed locally known as the "Big Seam," a component of the Clinton formation of Silurian Age. At present mining rates, this ore will be available for a number of years. It is largely mined under 1000 to 2000 ft of cover and transported on 15 to 20° slopes that, in some instances, are as much as 10,000 ft long. Under these conditions, particularly with increasing distances from face to portal, mining costs are gradually becoming disproportionately high. Although not a major factor thus far, it should also be noted that the average annual ore output of the district is gradually becoming more siliceous.

What, then, are the prospects for the future? Undoubtedly, large quantities of rich ore will be imported from newly developed South American and South African deposits, via the Gulf and our inland waterways. Such importations have already been used in the district to improve the grade of blast furnace feeds. The practice will undoubtedly be expanded. Concurrently, as mining costs on the Big Seam red ores increase with depth, there is little doubt that some method or combination of methods of concentration will be applied to the many millions of presently submarginal ores known to occur in the district.

The submarginal ores of the district consist of (1) several individual beds of ferruginous sandstones and weathered red ore outcrops in the immediate vicinity of Birmingham and extending northeasterly for considerable distances in a semicontinuous fashion; (2) beds of red ore similar to the Big Seam—in most instances too thin for a separate mining operation—but usually adjacent to the ferruginous sandstone mentioned above and probably best treated with them; (3) in some localities, 4 to 8 ft of lower-bench Big

Seam ore has been left in the red ore mines owing to a slightly excessive silica content. Part of the lower bench ore could be mined, part could not, owing to rock falls in the old stopes. In 1910 Burchard[1] estimated that the ore reserves in the lower bench of the Big Seam totaled 146 million tons. In the main, it has never been mined.

Probably the major consideration that will tend to make treatment of the low grade ores competitive is the fact that large tonnages of these submarginal materials can be mined inexpensively by open-pit methods. The fact that the beneficiated products will have a higher iron content than the ore "furnaced" at present will increase furnace capacity, as well as decreasing coke requirements.

This article is limited to a summary of the research work on Alabama iron ores undertaken by the Bureau of Mines. Nevertheless, recognition is accorded the excellent work of various steel-company research men. Coghill and Coe, in a bulletin that rather completely discusses the general history of iron making and research work on ore treatment in Alabama, frequently refer to numerous private investigators who have studied the ores. In a recent article Rose has discussed both Bureau of Mines and steel-company studies on the beneficiation of southern iron ores.

An intermittent study of various phases of the southern iron ore problem has been conducted at the Southern Experiment Station of the Bureau of Mines, Tuscaloosa, Ala., for about 30 years and a relatively complete bibliography is provided for those who may wish additional information.

Fundamental Studies

Singewald, Crane, Coghill, and Coghill and Coe made comprehensive studies of the location and grade of all six recognized iron-bearing strata of the Birmingham district over a linear distance ranging 25 miles southwestwardly from the Sadler Gap-Ruffner area. Their reports outlined variations in the iron content of the ore as it occurred in vertical sections at numerous localities and discussed variations in the grain size of gangue minerals contained in the different ore beds in the district. Numerous dissections of samples were presented and discussed which showed:

(1) The hematite of the southern ores is basically an amorphous form of the mineral that tends to form slime on grinding. In practice, only 10 to 20 pct of the hematite of the ores slimes rapidly upon wetting and light grinding. (Slime is defined as —20-micron material.) The remainder is reduced to slime more readily than a specular-type hematite but in most instances is more resistant to further size reduction than might be expected. As a result, a major part of the hematite in the red ores can be treated by hydraulic classification and tabling without excessive slime formation. (2) The dissection showed further that the ultimate grade of concentrate obtainable by any physical method is limited to 50 to 60 pct Fe by the presence of 8 to 15 pct of insoluble matter derived from an argillaceous diluent that is so closely associated with the hematite as to be physically nonrejectable. This does not mean that some higher grade mineral is not present—the writer has isolated hematite from Birmingham ores that contained as much as 65 pct Fe—but rather that, when recovery and grade are considered, most optimum-grade concentrates will have analyses of 50 to 55 pct Fe, regardless of the nature of the beneficiation method used.

(3) The studies showed further that appreciable quantities of hematite in ores from the Big, Ida, and Irondale seams are liberated in each screen size below 20 mesh. In these fractions there is also present true tailing and, in coarser sizes, a large amount of true middling. The quantity of middling decreases rapidly below 48 to 65 mesh, but some locking persists into —200 mesh sizes. The liberated hematite present in the coarser sizes is of as good quality as can be obtained by any reasonable degree of additional grinding. In general, the same conditions prevail in the case of the ferruginous sandstones, except that little hematite is liberated until ores are crushed to pass 35 mesh. Liberation ordinarily is attained with greater difficulty in treating the sandstones because a large proportion of the hematite is present as rather tightly bound shell-like coatings of the individual quartz grains rather than as oolites or fossils of tangible size and thickness. (4) Finally, the studies show that little reduction in the phosphorus content of iron ore concentrates is to be expected. If anything, the concentrated product will actually contain a slightly higher percentage of phosphorus owing to the rejection of gangue that contains no phosphorus.

From these fundamental studies, the investigators concluded that the ores should respond to concentration by either magnetic separation (after roasting) or by gravity concentration. Test work over the years has been directed toward a canvass of these and other possibly applicable processes.

Magnetic Separation

In 1927 Lee, Gandrud, and DeVaney published the results of a study by batch and pilot-plant methods of the roasting and magnetic concentration of samples of red ore from the Ida, Big, and Irondale seams. One sample of ferruginous sandstones was also included. The data showed that the ores were readily beneficiated by magnetic methods. The main question then, as now, being whether the over-all cost of magnetic roasting and subsequent concentrate sintering would be low enough for the beneficiated product to compete with natural ore at the furnace. Various attempts to apply the process commercially have not as yet been successful, and present high fuel costs make the future of the process even more obscure. Unquestionably, the method is attractive, primarily because (1) actual separation costs are low, and (2) it offers a means of at least partly relieving the difficulties of handling slime products. The thickening of hematite slimes would be eliminated, and filtration of magnetic concentrates in the manner practiced in the Adirondack mills would undoubtedly be a less difficult operation than the dewatering and filtration of hematite slimes.

Conversely, the process has certain disadvantages as applied to the southern ores. For example, in view of the inherent insoluble-matter content of concentrates from these ores, it would appear illogical to reject the calcite naturally

contained in some of the ores when lump calcite or dolomite must subsequently be added to the blast furnace charge. Fluxing stone is inexpensive, but the additional advantage to be gained from an intimate admixture of ore and flux would be lost. Lee's method of floating calcite from the magnetic tailing could be employed but would involve additional costs. Second, sintering machine operators prefer to treat feeds in which at least a part of the ore is moderately granular, whereas magnetic separation of red ores would require reduction of feeds to at least 100 to 150 mesh to avoid entrapment of true middling. Other weak points exist but need not be cited at this time.

Differential Grinding and Gravity Concentration

The presence in —20 mesh red ores of a moderately large amount of liberated hematite that could be recovered by concentrating tables led to a rather thorough study of gravity methods of treating the ores.

Much of the early work by DeVaney and associates, Gandrud and associates, and Coghill and Delano was directed toward establishment of conditions that would liberate hematite with minimum grinding of quartz. At that time this was a must, because there was no known method of removing ground quartz from the slime or fine hematite product that of necessity was accepted as concentrate. Their work established the practicality of beneficiating the high silica red ores by hydraulic classification and tabling, even though the procedures were based on an undergrind of the ores and recirculation of a heavy middling fraction through the grinding section. With the development of the flotation method of rejecting fine quartz from hematite ore slimes, to be described later, extreme care in grinding was no longer an absolute necessity, even though still desirable.

Coghill and associates, in their work on the red ores, studied the principles of classification and tabling as applied to these materials. Feld and associates developed a control method of rapidly estimating the grade of table concentrate. Clemmons and associates studied the possibilities of heavy-media separation of red ores from a majority of the mines operating in the district and found that the method could be economically applied to reject accidental contamination derived from parting-seam and rock-fall material whenever the latter were present in quantities as great as about 10 pct of the feed. The process was inoperable as a means of reducing the natural quartz content of the ores, due to the fine size of this gangue.

Clemmons and associates studied application of the Humphreys spiral concentrator to the gravity treatment of Birmingham red ores and found that, although the unit is not an all-purpose device permitting complete treatment of the ores, there is a good possibility that the unit will find application as an adjunctive beneficiation device.

Flotation

Clemmer and associates in 1945 reported successful solution of the problem of rejecting fine gangue from ball milled red iron ore slimes, particularly calcareous red ores. Their preferred procedure employed flotation of the naturally activated quartz (activated by lime salts present in calcareous ores) from strongly caustic alkaline pulps, while retarding the hematite into the flotation cell underflow. The quartz was filmed and floated by means of tall oil, an inexpensive paper-mill waste product. Several patents granted on processes developed in the work provide additional information on the separation. Typical results obtained on Big Seam ore from five Birmingham mines are presented in Table I. Reagent costs amounted to 14¢ per ton of feed or 17 to 20¢ per ton of concentrate. Workability of the process was thoroughly demonstrated by operation of a pilot-plant flotation unit in a Birmingham concentrator for several weeks.

In recent years studies have been continued and expanded to include the noncalcareous surface (leached) red ores and the ferruginous sandstones. Concentration of these materials necessitated preflotation treatment of the ground ore pulps to activate the quartz for subsequent flotation and rejection. Use of the process in batch tests made on naturally leached red ores yielded products containing from 50 to 59 pct Fe, depending on the quantity of argillaceous gangue present in the ore.

Table I.—Typical Results Obtained by Floating Quartz From Commercial Red Iron Ore

	Mine run Feed, assay pct			Iron flotation product			Iron flotation product, lime free basis*		Recovery, pct
Mine	Fe	CaO	Insol	Fe	CaO	Insol	Fe	Insol	Fe
A	34.7	14.6	19.2	40.7	14.4	10.8	54.8	14.5	96.0
B	35.8	13.5	19.7	42.2	12.6	11.8	54.5	15.2	95.5
C	39.7	10.6	19.7	46.7	10.3	10.0	57.2	12.2	95.6
D	38.6	12.7	17.5	45.3	12.1	8.7	57.8	11.1	94.5
E	35.8	12.5	22.0	44.5	12.1	9.8	56.8	12.5	92.4

* Lime content of iron product calculated out. Results permit better comparison with Mesabi ores and also with results that might be obtained by magnetic separation. Actually, rejection of calcite was not sought, as it was preferred that the material be retained with hematite for fluxing purposes.

Samples A and B contained some shale, which was not rejected by the process.

In a recent cooperative study with the Sloss-Sheffield Steel & Iron Co., activation processes formerly used only in batch testing have been successfully adapted to a continuous operation pilot-plant scale. Pilot-plant tests were made on a 28 to 29 pct Fe sample from a vertical 64-ft outcrop (containing Ida, fine-grained Big Seam, ferruginous sandstone, and Irondale ores) in the northeast Birmingham area. A combination of classification, tabling, and flotation was used, resulting in iron products containing 49 to 51 pct Fe, with 85 to 90 pct Fe recoveries. Determination of the true quartz contents of the iron products showed that about 90 pct of the quantity of this mineral present in the feed had been rejected. The remaining 18 to 19 pct insoluble content of the concentrate was largely due to nonrejectable argillaceous material. Reagent costs amounted to 25 to 30¢ per ton of concentrate.

Similar tests, made on a friable, crumbly, surface ore from the area (again a mixture of ore beds, containing only 23 pct iron), resulted in the recovery of a 50.5 pct Fe product. Reagent costs were 13¢ per ton of feed or 36¢ per ton of concentrate.

In the combination gravity-flotation process, selective grinding continues to be desirable but is no longer a necessity. The grind is held as coarse as is commensurate with the liberation characteristics of the ore being treated. Hydraulic classification is not highly critical, and separation into two or three sizes in all likelihood will suffice. Thickening and filtration of fine material are probably the greatest problems involved in the process, but are not insurmountable difficulties, due to recent technical advances in the art. Part of the thickening difficulties in the district, in the past, has been due to the use of the thickeners as "storage bins." An investigation of the possible utilization of Netherlands State Mines cyclones as a means of improving the situation is now in progress at this laboratory. Dewatering by centrifuging may also find application in the work.

Conclusions

What, then, is the best method of concentration for the low-grade hematite ores of the district? A good brief, presented by Rose, for magnetic separation, following reduction roasting by fluidized solids methods, has several attractive features. All magnetic concentrate would have to be accepted at —100 or 150 mesh owing to the locking characteristics of the ores.

Conversely, by the combination gravity-flotation concentration method, many of the ores will permit recovery of appreciable amounts of hematite in sizes up to 20 mesh and concurrent rejection of a large amount of natural-size quartz. Pilot-plant tests have shown that 40 to 60 pct by weight of plant feeds can be rejected on concentrating tables and need not be sent to flotation (table concentrates contain 50 to 55 pct Fe and table tailings 4 to 12 pct Fe). This should reduce grinding costs considerably. Flotation-reagent combinations have been developed and verified on a pilot-plant scale that permit ready treatment of the entire slime fraction of the ores whenever necessary. The process allows rejection of essentially all quartz gangue from fractions as fine as 5 microns in size. Methods have been devised that permit either removal of calcite as a separate product or retardation of the mineral with the iron, as desired. Hematite slimes have concurrently been thickened and as much as 75 pct of the mill water recirculated for considerable periods without deleterious effect. Pilot-plant results (which experience has often shown indicate higher than commercial plant costs) show that the cost of reagents will be 20 to 25¢ per ton of ore treated, depending on the nature of the ore being concentrated. In view of these and other considerations, it is believed at this time that the described combination process, or some variation thereof, offers the greatest chance of economic application in the district.

Regardless of the concentration method used, some means must be provided of agglomerating the product for blast furnace use. Various methods have been employed in the district, but no correlation or comparison of the results of the several methods has been attempted. The Bureau of Mines hopes to undertake studies of this type at an early date.

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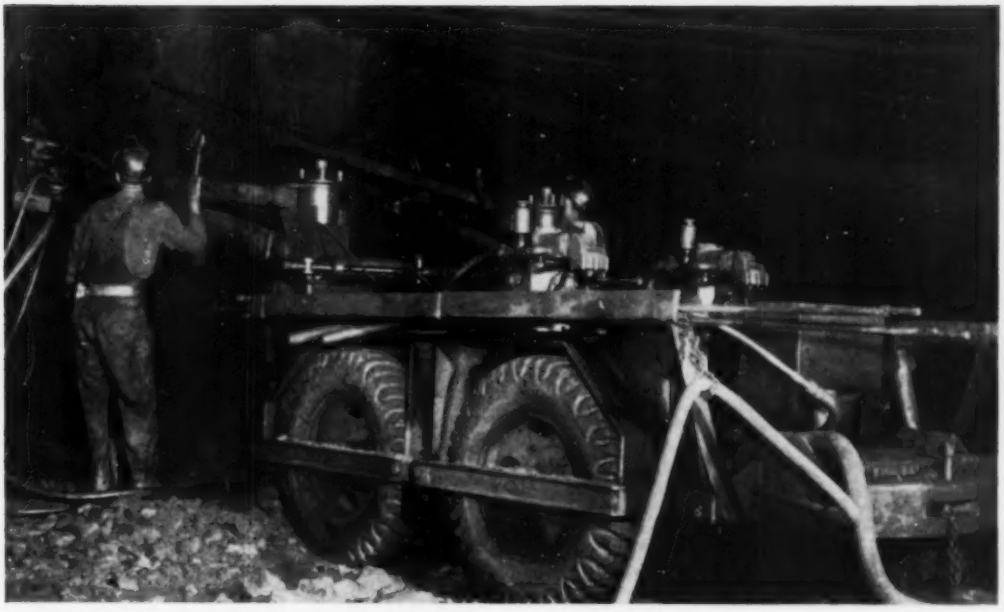
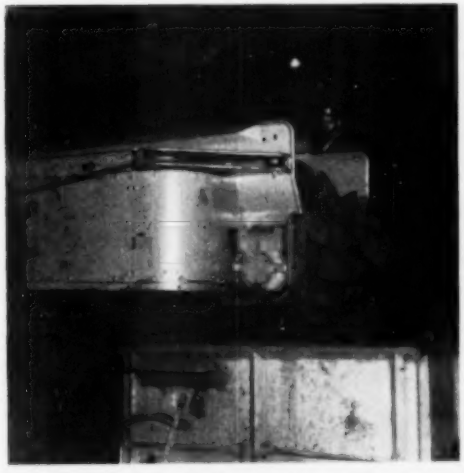
    (All patents assigned to the United States of America as represented by the Secretary of the Interior.)



Trackless Mining
Improves Ore Production for TCI

by Leland H. Johnson

assistant chief engineer, ore mines and quarries, Tennessee Coal, Iron & Railroad Co.



Above, a 14 ton capacity shuttle car discharges into a 10-ton mine car. At top of page, one of the air-operated, rubber-tired drill carriages used in trackless mining sections is drilling in room face.

THE Tennessee Coal, Iron & Railroad Co.'s ore mines began experimental tests with trackless mining units early in 1947. The units had many weaknesses to overcome, but are now well beyond the experimental stage and are becoming a necessity to ore seams that are amenable to this type of equipment. The development of this type of mining with the problems of increased underground supervision, maintenance, and training personnel, along with the benefits derived, are discussed.

The company owns and operates a group of eight iron mines along the crest of Red Mountain between Birmingham and Bessemer, Ala. These mines produce about 5,500,000 tons of ore a year with an iron content of about 37 pct. The Big Seam, the only ore bed of commercial importance, extends along the outcrop for approximately 15 miles and dips to the southeast from a few degrees to as much as 30°. It ranges in thickness from 10 ft to 16 ft.

Slopes are driven down the dip from the outcrop to a depth of from 5000 to 8000 feet. Electric hoists are installed on all slopes except two, which are equipped with steam hoists. Storage pockets are placed at the underground rotary

Electrically-powered crawler mounted loader, with a capacity of 8 to 10 tons per min, discharging ore into a shuttle car. Twelve more of these machines will be placed in operation at TCI mines during the first quarter of 1951.

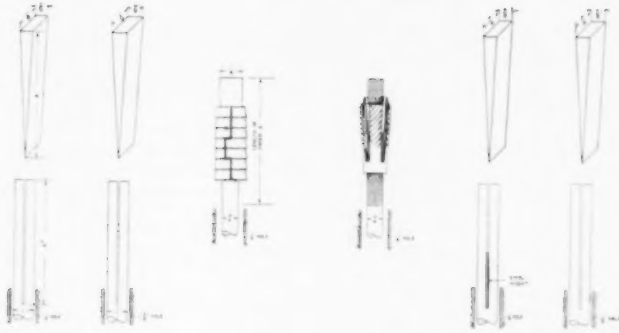
dump locations. The ore is drawn from the pockets into measuring pockets and thence dumped into 15-ton skips and hoisted to the surface. At the surface, the skips dump automatically into 70-ton capacity railroad cars, and thence are hauled to a central ore conditioning, sintering, and briquetting plant.

Prior to 1923 all ore was loaded by hand. The first double-drum scraper loaders were installed in 1923 and all the mines were equipped by 1925. The first track-mounted loaders were placed in operation in 1942 and were air-powered. Electrically-powered, track-mounted loaders were tried experimentally in 1947 and additional units were obtained in 1949.

In February 1947, an electrically-powered, crawler-mounted loader was placed in operation at No. 11 mine on an experimental basis. This loader is equipped with two rotating gathering arms which crowd ore to a boom conveyor for loading into rubber-tired shuttle cars. This machine was a modification of a conventional loader used in coal mines and was strengthened to handle the heavy iron ore. Several weaknesses in design developed and a new loader was placed in operation in September 1947. Separate motors were utilized for the conveyor drive. Improvements were made on the gathering arms and clutches, and a wider gathering head was installed. Experience with this model over a period of two years indicated a need for further ruggedness in order to reduce maintenance costs. In 1949, three machines of an improved design were placed in operation, making a total of five in use. They were equipped with longer crawler frames, wider and heavier shoes, individual crawler control, heavier conveyor chains with individual higher torque motors, heavier bearings and supports for gathering-arm discs, heavier drive chains for gathering arms, and other improvements of a minor nature. Twelve similar machines are on order and will be placed in operation during the latter part of 1950 and in the first quarter of 1951. The overall length of the machine is 25 ft 8¼ in.; the overall width is 6 ft 6½ in.; and the overall height is 5 ft 5½ in.

The loading capacity is 8 to 10 tons per min. In 1949, the panel loaders averaged 425 tons per

One of 6 air-propelled crawler-mounted units used to drill roof-bolt holes and to drive and tighten bolts. These were built at TCI machine shops from crawler mounts and with installation of motors and hydraulically raised platforms.



Factors considered in these roof bolts were bolt diameter, spacing, hole diameter, wedge size, design for anchorage and bolt strengths.

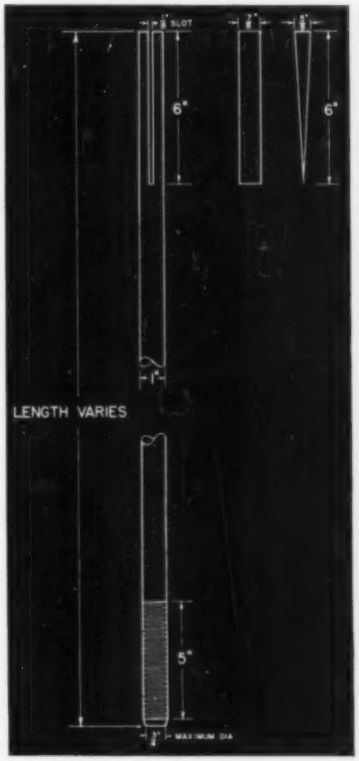
turn and during the first six months of 1950, the average has increased to 502 tons per turn. In robbing operations, the average during the first six months of 1950 was 514 tons per turn. The highest weekly average on a machine is 748 tons per turn and the highest production on any single turn was 920 tons. The increase in tonnage is primarily due to a decrease in delay time. In 1949, the delay time averaged 9 hrs per week and during the first six months of 1950, the delay time has dropped to 7.6 hr per week. With the recent increase in maintenance personnel, it is expected that delay time will continue to decrease with a subsequent increase in production.

An electrically-powered, crawler-mounted loader of another manufacture was received in September 1950, for tests in loading iron ore. This machine has a bucket that pushes into the muckpile, and discharges over the machine with a rocker-arm mechanism. Data are not yet available on performance, but it is estimated that it will load at the rate of 5 to 6 tons per min. The overall dimensions are as follows: length, 13 ft 8 in.; width, 5 ft 10 in.; and height, 5 ft 3 in.

Shuttle cars are in use for conveying ore from the trackless loaders for discharge into mine cars. These cars have a capacity of approximately 14 tons, and are 24 ft long, 8 ft wide, and 4½ ft high. The road clearance is 12 in. and the travel speed is 3 mph empty and 2 mph loaded. The drive is on the rear wheels only and the steering is accomplished hydraulically on the front wheels. The conveyor speed for unloading is 55 fpm. Ten shuttle cars are in use and 24 are on order.

Air-operated, rubber-tired drill carriages are in use on trackless mining sections. Two 3-in. cylinder drills with 6-ft chain feed are mounted on movable arms on these jumbos. Hydraulic controls are utilized for positioning the drills. The working range for the jumbo is 10 to 12 ft in height and 20 to 22 ft in width. The overall dimensions are 22 ft long, 8 ft 1 in. wide, and a wheel base of 4 ft. Two jumbos are utilized for each panel section and the average number of faces drilled and blasted is 2.7 per turn. The average advance is 6.5 ft per round using a 34-hole, double-v round. The average amount of ore broken per blast was 192 tons during the first six months of 1950. Several types of steel, with and without bits, have been tested in the mobile sections and the best results obtained were with 1-in. and 1⅛-in. alloy rods having tungsten-carbide chisel inserts. Drilling speeds in iron ore with this equipment have exceeded 30 ipm.

The support of roof in all trackless mining sections is accomplished with roof bolting. An air-propelled, crawler-mounted unit with a lifting platform is utilized to drill the holes and to drive and tighten the bolts. The units have been constructed in the ore mines and quarries machine shop from crawler mounts less engines and with the installation of air motors and hydraulically raised platforms. The platforms are 10x6 ft in size, with a guard rail, and racks and hangers for the roof bolting equipment and supplies. Opposite page shows one of six now in use. Ten more of a similar design will soon be constructed.



Type of bolt selected for use was this 1-in. round rivet steel bolt with burned 6-in. slot and 5-in. of cut thread.

Roof bolting was started in the ore mines on July 6, 1948 and rapidly expanded to all mines. No other method has been used for more than a year and over 6 million sq ft of roof is supported entirely by roof bolting. Stated in terms of lineal footage along an entry 20 ft wide, there are now about 60 miles of bolted roof.

Continuous testing and experimentation has been required to obtain roof control, and this work is still going on. The types of bolts tested are shown at top of p. 1227. Such items as bolt diameter, spacing of bolts, hole diameter, wedge size, bolt design for anchorage and bearing and roof loads and bolt strengths are vital factors for obtaining adequate control. Each of the factors had to be considered in relation to the others and in relation to the many variables in roof conditions that cannot be determined mathematically.

These tests have led to the ore mines' adoption of 1¼ in. hole diam and 1-in. round rivet steel bolt with a burned 6-in. slot and 5 in. of cut thread. The steel wedge has a ⅝-in. thick x ⅞-in. width x 6-in. length. The bolts vary in length from 5 to 8 ft according to the determination of diamond drill data. Accessory equipment was developed, such as driving dollies, bearing plates, angle washers, and channels. These items were modified as experience dictated. The driving dollies are designed to deliver the stoper blow to the bottom of the bolt, not to the nut or threads. The bearing plates are 8x8x½-in., and have a 1⅛-in. round hole for vertical bolts and an elongated hole for use on a 60° angle rib bolt in conjunction with a 3x2x½-in. angle washer. Where the roof strata are broken or cracked and it is desirable to spread the bearing of the roof bolt support, prepunched 5-in. channels in lengths of 14, 16, 18, and 20 ft are used.

Bolts are installed on 5-ft centers, are driven for at least 20 sec, and the nuts are tightened to obtain 400 to 450 ft-lb torque. Bolts are placed within 2 ft of the face prior to blasting.

Roof bolting has to a large degree been responsible for the following advantages not present in the older methods of timbering in trackless mining sections: (1) Tonnage from all mines has shown a steady increase. (2) Safety experience for the year 1949 was the best in the history of the ore mines with a frequency rate of 7.83 injuries for each million man-hr of exposure. This was 77 pct lower than the 1948 U. S. Bureau of Mines frequency rate for underground mines, which was 29.31 injuries per million man-hr worked. (3) Roof bolting eliminates a laborious job of placing conventional timbering. (4) Less quarrying has to be done. (5) There is an increase in vertical clearance. (6) Better ventilation is accomplished through unobstructed airways. (7) There is a lower direct timbering cost. (8) Re-setting timbers has been eliminated. (9) The quality of the ore has improved.

Electrically-operated bulldozers are also in use underground for speeding up track grading on main haulages, for hauling ties and rails, for maintaining clean and smooth travelways, and for bulldozing scattered ore into piles for more efficient use of loading equipment. Two standard bulldozers equipped with 35 hp d-c electric motors, motor-driven cable reels, and 500 ft of cable, and weighing approximately 10,000 lb have been in use since October 1948. Experience has indicated that a heavier machine would be more desirable and two are now on order that will weigh approximately 13,250 lb. The blade length is 7 ft 2½in.; the overall height is 5 ft ⅝ in.; overall length is 12 ft; and the overall width is 7 ft 2½ in.

A complete change in mine layout was necessary for mechanized mining. A system was adopted with 4-entry main haulages, 3-entry panels, and 22-ft rooms that vary in length from 300 to 500 ft. Uniform grades and right angle rooms off the panel haulage are not possible due to the various dips encountered in the ore seam. Diamond drilling is carried on in development headings to determine the best possible grade that can be utilized. The average grade on haulages is about 4½ pct against the loads and the panel rooms are laid out so that the grades will not exceed a maximum of 7 pct in favor of the loads.

Loading ramps constructed of 3x12-in. planks are used for shuttle car discharge into 6 and 10-ton mine cars. Four and 5-car trips are shifted at the loading ramp by either locomotives or remote control hoists. Two elevating conveyors are available for loading mine cars and three more are on order. These machines are used when the ore seam becomes too thin for proper and safe clearance at normal ramp discharge locations.

The mining plan has varied with relation to pillar sizes and crosscut intervals. The first haulages were driven on 150-ft crosscut centers and 50-ft pillar widths. This plan has been modified to 100-ft centers and 30-ft pillar widths in recent layouts. On panels, the original rooms were mined on 110-ft crosscut centers and 50-ft pillar widths. Recent experience on pillar robbing indicated that the pillars should be smaller and layouts now show crosscuts on 60-ft centers with a pillar width of 26 ft. This allows 60 pct extraction on first mining with a safety factor of 3.0 and 80 pct extraction on second mining or robbing with a safety factor of 1.5. The safety factor is based upon the depth of cover and is computed from a formula that takes into consideration the crushing strength of the ore and the weight of the overlying cover. The crushing strength of the ore is 13,000 psi.

Ventilation and dust control are more easily controlled in trackless mining sections. Each panel is on a separate split of air ranging from 25,000 to 30,000 cfm. This volume is drawn in the haulage and manway and is directed to the last open crosscut from whence it returns on a lower aircourse. Four auxiliary fans of 6000 cfm capacity and 18-in. flexible tubing are used to carry fresh air to the face of each working place in a panel. All loading equipment is equipped with water sprays, and a separate hose with a full-cone spray nozzle is available at each working face for wetting the face, ribs, floor, and roof before and after blasting.

The trackless mining sections show a better quality of ore. In drag or slusher mining, the stopes ranged in width from 110 ft to 160 ft and were 210 ft in length. The greater pressures which result from deeper cover have caused ledges of slate and sandstone overlying the ore to break more readily. This foreign material causes dilution and lowers the quality of the product shipped. The trackless mining system

affords better control of roof and dilution is held to a minimum. Also, the use of drill jumbos allows for holes to be drilled parallel to the roof and blasting does not cause the shattering of the rock ledges overlying the ore.

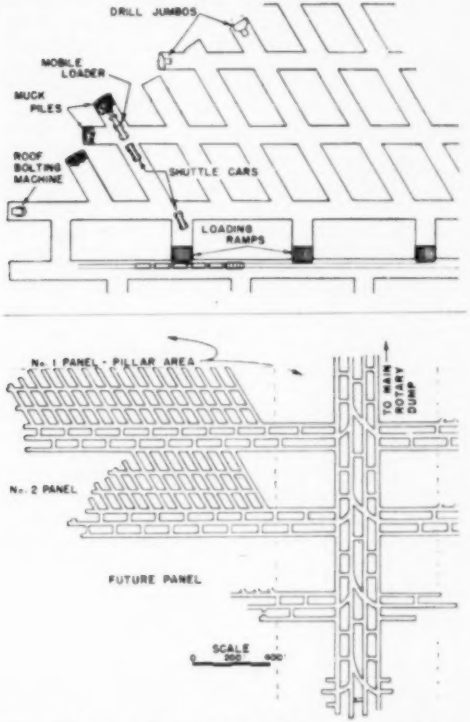
One of the most important factors in the success of trackless mining has been the training of employees to operate and maintain the equipment. In November 1947, a mechanical mining team was organized to assist in training the men employed in new sections. It was composed of a mine captain, two assistant mine foremen, two industrial engineers, and a general maintenance foreman. Training of the team covered a period of two months and included instruction by the Tennessee Co. Training Bureau, inspection trips to other coal and limestone mines and the inspection of manufacturing plants that make the trackless equipment. The team is constantly working to improve units already in operation as well as training men for new units. A series of 2-hr lectures totaling 24 hr was prepared for training and 171 men including supervisors and engineers attended voluntarily with a 92 pct attendance record. A handbook of mechanical mining was prepared by the engineering and operating staffs of the ore mines and quarries division. A second edition was recently prepared and it is used extensively to train foremen and head miners whenever a new unit is placed in operation and to establish uniform procedures in units already in operation. The handbook covers such topics as loading, roof bolting, drilling and blasting, quarrying, supply, electrical layouts, care and maintenance, job breakdowns, operating and maintenance reports, employment of crews during cycle delays, basic organization chart for section crews, and layouts of driving widths. Compressed air lines, water lines, power lines, track, hoist, and ramp layout, drainage plan and ventilation plans are also included.

Extensive courses have been given to maintenance personnel covering the fundamentals of mechanics and electricity. The educational program of correspondence schools has been utilized a great deal.

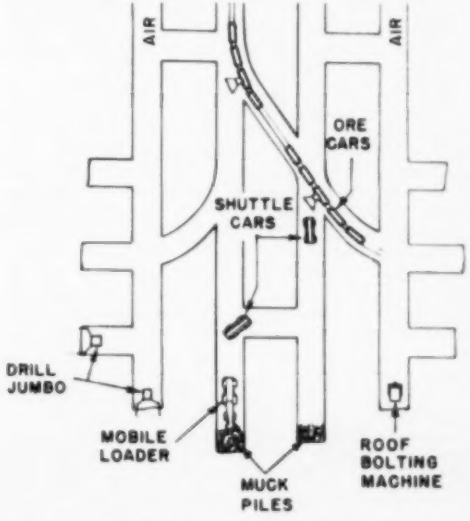
In order to make a trackless mining system successful, the loading machine must be kept loading ore as continuously as possible. All other operations such as drilling, blasting and roof bolting are keyed to a definite cycle to keep the loader operating. The cycle used in ore mines is drill and blast, then load, and finally roof bolt. Close and thorough supervision is necessary to obtain a cycle that will function properly and provide a plentiful supply of broken ore to load.

The results of trackless mining to date are quite encouraging and present plans are to expand its use as rapidly as possible. Direct mining costs have shown reductions of approximately 12 pct on panel mining and 20 pct in development mining when compared to the use of drag scrapers and track-mounted development units, respectively. The production rate on producing employees has only increased from approximately 14 tons per man-turn to 23 tons per man-turn. The safety records show a decided decline in numbers of accidents and this record can be attributed in part to the change to trackless mining. Ventilation is far better controlled and affords larger volumes of air to the active working faces.

The management of the Tennessee Coal, Iron & Railroad Co. has realized the potential benefits that complete mechanization with trackless units will bring and every effort is being made toward achieving this program of mining.



The multiple-entry system, formerly distinctive of coal mining is used effectively by TCI in the ore mines. These figures show a general plan of panel development (lower left) and details of the mining cycle (upper left) and the sequence of operations in developing entries (right).



Opening the Pyne Mine of

by John V. Beall, editor, Mining Engineering

THIS is not simply the story of how a water filled shaft was developed into a million-ton-a-year producing mine in the space of four critical years, although it is reason enough for telling it, but it is also the saga of an engineering achievement in which knowledge nurtured by years of experience in coal and ore mines was blended together and the best from each type of mining selected. The entire system of mining was detailed on maps by the mining engineering department before the shaft was unwatered, and subsequent development followed these plans nearly to perfection. The selection of machinery was particularly critical as it was necessary to push production from only a few headings in the early stages of development and metal mine machinery was being used in a coal panel-system of mining.

At the start of World War II the demand for iron ore to supply Woodward blast furnaces, and rapid exhaustion of the main source of Woodward ore, No. 3 Red Ore mine, necessitated speedy opening of the Pyne mine. The mine is three miles southwest of Bessemer and is responsible for 80 pct of Woodward ore, the rest coming from the Songo mine.

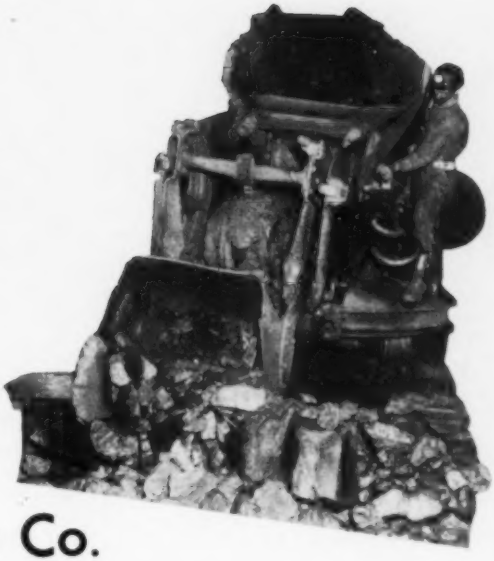
The Woodward Iron Co. is one of the old producers of the Birmingham district. S. H. Woodward, an iron-master of Wheeling, overheard two Union soldiers conversing about ore and coal lands in Alabama, which lead to the acquisition of the first ore lands of the Woodward Co. in 1869. Development was not begun until after the death of S. H. Woodward; his two sons, W. H. Woodward and J. H. Woodward, formed the Woodward Iron Co. in 1881 and purchased a tract of land midway between the ore vein on the southeast and the coal field on the northeast about 12 miles southeast of Birmingham for the erection of blast furnaces and coke ovens. The brothers brought with them $400,000 in cash with which to build a blast furnace, 150 beehive ovens, open both ore and coal mines, and construct about eight miles of railroad. In 1887, with two furnaces each producing about 165 tons of pig iron daily, the ore cost was 68¢ a ton delivered and coal 80¢ a ton. Since the first furnace was blown in in 1883, iron has been produced without interruption, the company making more than 18 million gross tons of pig iron. The company has sufficient reserves of coal and iron ore to operate its three blast furnaces, and the new one presently under construction, at peak capacity for at least the balance of this century. The Woodward Iron Co. ranks today as the largest independent and completely integrated manufacturer of merchant pig iron in the United States.

Before World War II, plans had been made for the opening of the Pyne mine and a development procedure decided upon which would yield sufficient production to offset the expected decrease in production from the No. 3 Red Ore mine. Pearl Harbor changed the picture at Woodward as it did in every manufacturing industry. More ore was needed, and quickly. The Pyne mine was, in 1942, a 1300-ft water-filled shaft. Work com-

Table I
Yearly Production Record of the Pyne Mine

	Long Tons
1942	32,201
1943	186,379
1944	510,110
1945	902,856
1946	1,109,942
1947	1,208,175
1948	1,147,702
1949	1,053,624
Total	6,150,989

AN enviable record was established in the speed with which the Pyne mine was opened. Today it produces a million tons annually using the coal panel system of mining. This is the first story of how the mine was brought into production, and the mining system.



the Woodward Iron Co.

menced with bailing the shaft with the ore skips. Fig. 1 is a plan view of the method of development as it was finally carried out and it differs from the original plans only in that the stope headings at the west end of the property numbered 104, 105, 106, etc. were driven on the oblique to take advantage of a more favorable slope. Otherwise the development was driven as originally laid out. At the start of development in 1942, only one working face driving north from the shaft was available. As this drive was branched at the shaft yards two headings were worked and as the original headings were advanced more branches were started and tonnage was gradually built up.

Although it was originally planned to drive headings to the property limits and retreat toward the shaft with a 45° face angle, the need for faster production dictated the opening of the 400 panel, the nearest immediately east of the shaft. The production record shown in Table I is graphic testimony of the effectiveness of the mining method, the equipment used, and the men behind the machines. The first ore was hoisted in July 1942 and came from the one working face at the foot of the shaft.

Today it is probable that the Pyne mine produces the greatest tonnage through a single shaft of any iron mine in the United States.

The upper bench of the Big Seam is being mined and averages 10½ ft thickness. It is a rolling bed making grades up to 20 pct in some haulageways. Double entry haulageways were driven 8000 ft east to the property line and 3200 ft west from the shaft. Air courses are driven parallel to main haulageways and panel headings. Both ventilation and haulage headings are 12 ft wide

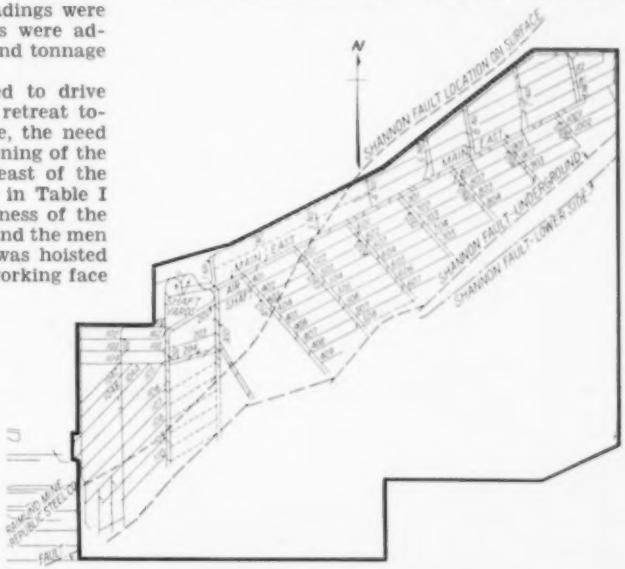


Fig. 1—Panel mining with the split system of ventilation at the Pyne mine, as shown, is flexible to systematic mining or quick tonnage.

x an average height of 10½ ft. Panel headings are turned off the main haulage on 1000 ft centers and driven south to the Shannon fault and north to the property line. In driving south to the fault which is an aquifer, diamond drill holes precede the heading so that the time for stopping without cutting the fault can be determined. Stope headings are turned off panel headings on 210-ft centers. Development work was performed with Woodward design jumbos on which were mounted power fed drifters. The machines have sliding cone shell with 30-in. feed; 1¼-in. steel and detachable bits are used. All types of bits have been, or are being, used including throwaway and carbide insert. Bonus is paid on top of base wage for linear advance in excess of that required for base wages. A heading crew comprises a head miner, 2 drillers, 2 helpers, motorman, coupler, 1 shovel operator, and helper who all share in the bonus on a scale proportionate to their job. Eimco 40 rocker shovels are used for cleanup. The speed with which development work progressed can be attributed to the use of these machines and to the adoption of a multiple heading development cycle. Multiple shifting of equipment in driving multiple headings is an old idea in coal mining in the Birmingham area but was not widely used in the ore mines. It was difficult at first to sell this method to the ore miners but once they caught on to it, it became popular because of the added income from greater footage.

Both single and double-wing stopes are turned off stope headings on 60-ft centers. A 100 ft pillar adjacent to the fault is left intact. In the next 400 ft, only 50 pct extraction is taken in single neck stopes on first mining. This permits adequate ground support in the uncertain area under the Shannon fault.

In the double-wing stopes two wings are driven, each 20-ft wide, with a 10-ft pillar between wings and stopes. Both the pillar on the inbye wing and the pillar between the wings are slabbed on retreat. Only a few small pillars are left of these 10-ft strips. About 80 pct extraction is obtained. Three double-wing stopes are usually worked per heading each loading on a separate shift. One drag hoist is installed for each double-wing stope. Ottumwa Iron Works double-drum hoists driven by 55 hp 230-v dc Crocker-Wheeler motors are used. The drags are mounted in cut-outs on the stope headings opposite the stopes and the ore is dragged up a ramp into cars on the heading by a 4 ton capacity drag scraper. Two shifts of drilling are required for each one of mucking. Drilling is done with drifters mounted on tripods. Roof bolts for roof support are used in 60 pct of the stoping operation. Rods 54 in. long are put up on 5-ft centers using an Inger-

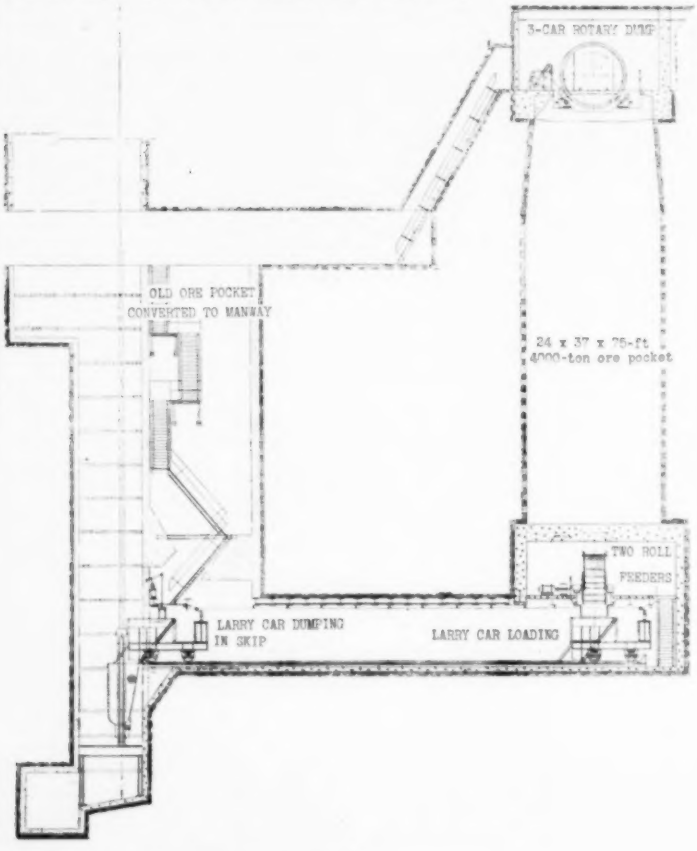
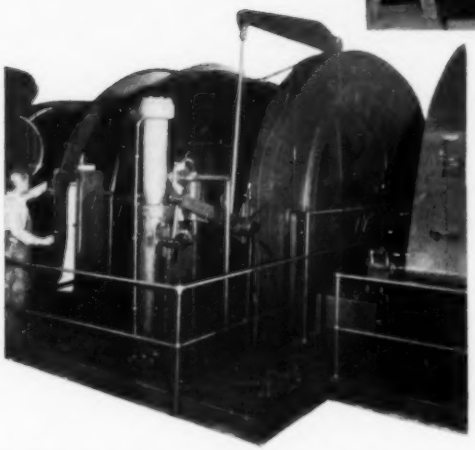

3-CAR ROTARY DUMP
OLD ORE POCKET CONVERTED TO MANWAY
24 x 37 x 75-ft 4000-ton ore pocket
TWO ROLL FEEDERS
LARRY CAR DUMPING IN SKIP
LARRY CAR LOADING

Fig. 2—Roll feeders take the ore from pocket to larry cars which dump into skips. An interlock prevents skip from being hoisted until larry car has moved back from skip.

At right the 3-car Roberts & Schaeffer rotary dump is unloading ore into the pocket. Below is a view of the Nordberg main ore hoist which has double drums, cylindro-conical cylindro.



soil-Rand impact wrench for tightening the nuts. Production from a double-wing stope averages 225-tons per day with a crew of 11 men, and 9500 tons is produced during the life of a stope.

One feature of the mining plan is that any panel can be worked without interfering with mining in any other panel. Barrier pillars are left so that haulageways, ventilation entries, and shafts are fully protected. The mining plan calls for retreating from the property limits to the shaft recovering the pillars as the entries go out of service.

In general, haulage is by electric locomotives on 42-in. gage track, with 80-lb rail on creosoted ties on the main line and 60-lb rail on stope headings. However, in some places the grade is so steep that hoists must be used instead of locomotives to haul the cars. Single-drum 55-hp 18x30-in. slow-speed entry hoists are used in these instances. Haulage is in rigid-body 10-ton cars. Locomotives range in size from 6-ton to 36-ton tandems on main line haulage. All haulage is on 230-v trolley.

At the shaft, ore is dumped by a 3-car Roberts & Schaeffer rotary dump. The ore pocket is 24x34x75 ft and holds approximately 4000 tons. The Pyne shaft had an old pocket which fed directly to the skip compartment but as it was too small the new pocket was constructed. Two roll feeders draw the ore from the pocket into 10-ton larry cars which operate on separate tracks between the pocket and the skips. These cars are self-propelled, by remote control and travel out over the skip to dump. The skips rest on buffer blocks while being loaded and the power to the hoist is automatically cut off while the larry car is over the skip. After the larry car finishes dumping and moves away from the shaft it automatically trips the power back on the skip hoist and the skip is hoisted in response to the loading pocket operator's signal. Loading the skips requires two men; one at the roll feeders and one at the shaft. The larry cars were designed and constructed at Woodward. Fig. 2 shows the layout of the ore pocket and skip loading arrangement.

Ore hoisting is in ten ton balanced skips. A Nordberg hoist of the following specifications is used:

Table II
Specifications of Nordberg Main Ore Hoist

Double drum- cylindro-conical-cylindro
2100 tons per 7-hr shift
15,300 lb weight of empty skip
22,400 lb weight of ore in skip
1¾-in. hoist rope
982 fpm max. rope speed
10 sec acceleration, retardation, and dumping time
12 ft diam drum at small end
16 ft diam drum at large end
14 ft diam post brakes
10 ft multiple tooth type clutch
Accumulator-Nordberg pressure type
800 hp GE synchronous motor
Automatic control GE

The main shaft is a 6-compartment vertical shaft, 1213 ft to the ore level and 1280 ft to the loading pocket. The shaft is approximately 20x12 ft inside steel; each of the two skip compartments being 6 ft 6 in. x 6 ft 3½ in., two men and materials compartments served by double-deck cages are 10 ft 6 in. x 6 ft 3½ in., and two manway and pipe compartments 3 ft 6¾ in. x 6 ft 3½

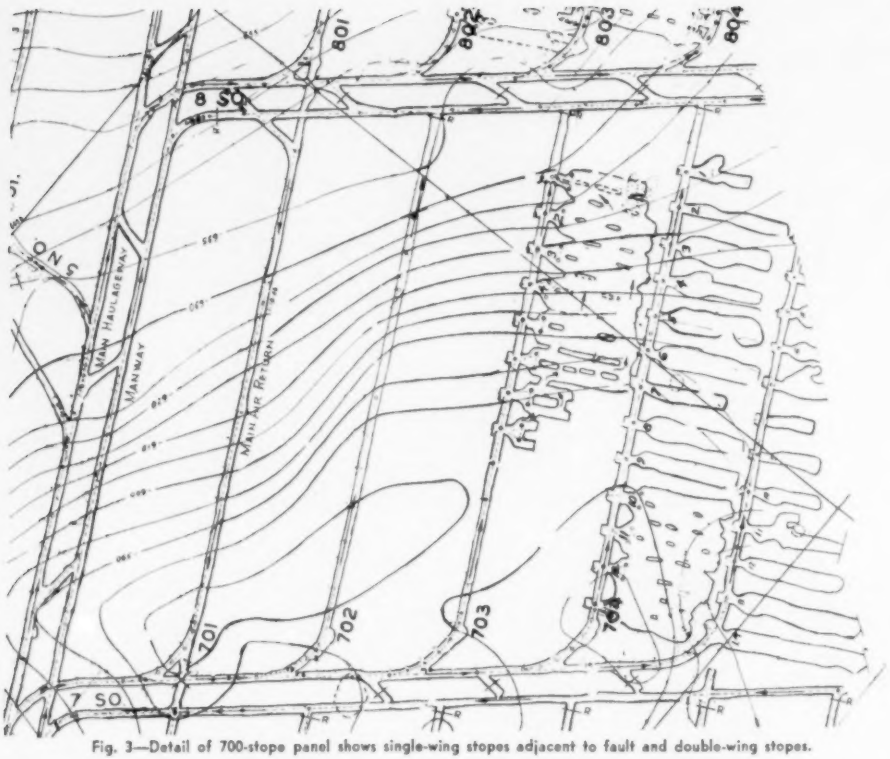


Fig. 3—Detail of 700-stope panel shows single-wing stopes adjacent to fault and double-wing stopes.

in. The man and material hoist is a 350 hp double cylindrical drum hoist, 8 ft diam, manufactured by Wellman-Seaver-Morgan, with Lilly control. A $1\frac{3}{4}$-in. rope is used.

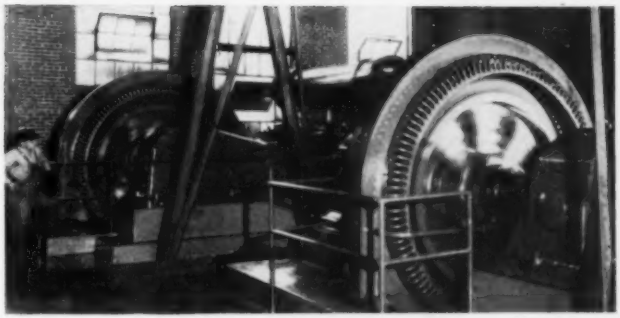
The ventilation system is the split system used widely in coal mining and adapted to ore mining. A concrete circular ventilation shaft, 14 ft ID, was started in 1942. This shaft was completed in 1943. The ventilation equipment consists of an 84-in. Jeffrey Aerodyne fan exhausting 145,000 cfm. It is mounted at the surface of the ventilation shaft. A 100-hp GE motor drives the fan which is rated at 300,000 cfm. Air comes down the main shaft, progresses along the main entry and is split at each working stope panel. Air from the panel headings is drawn off for each stope heading and is regulated in each stope heading return. Each split of air passes through a working stope area and is exhausted through the air course of the next panel heading into the main return air course (Figs. 1 and 3). Auxiliary blowers and air ducts are used in single development headings. The air shaft is equipped with a hoist and eliptical man cage which can be used as an escapeway.

The main pumping station is located near the foot of the shaft. It contains two 250-gpm I-R 6-stage centrifugal pumps driven by 125-hp GE motors and one 500-gpm pump driven by a 250-hp motor. They work against a 1240 ft head, discharging through a 12-in. pipe in the shaft.

The red brick surface buildings are of modern construction and include a 500-man bathhouse providing lockers for street clothes and baskets for work clothes in separate rooms with showers between. There is also a machine shop, a bit sharpening shop, and a blacksmith shop.

The compressor house contains two Hardie-Tynes horizontal compressors of 3000 cfm capacity driven by 500-hp GE motors. Power is generated at 44,000 v at Woodward and is received at a transformer station at the Pyne mine. There is also a power tie in with the Alabama Power Co. at Woodward.

Before the ore from the mine is loaded into railway cars for transportation to the furnace, it passes through the surface plant which includes two stages of crushing and a screening plant. Ore from the headframe bin is crushed to—6 in. in a 10-K Allis-Chalmers gyratory crusher. The dishcarge from the crusher is elevated on a 42-in. belt conveyor to the second stage of crushing. A scalping screen by-passes the —$2\frac{3}{4}$-in. material while the coarse is reduced in a No. 7 Symons cone crusher and then taken by belt conveyor to the screen plant. At the screening plant one 6x 14-ft Robbins double-deck vibrating screen produces three sized products for shipment. Fifty pct of the ore is lump, 3x$1\frac{1}{2}$ in.; 25 pct medium, $1\frac{1}{2}$x$\frac{3}{16}$; and 25 pct fine, —$\frac{3}{16}$. The fine ore is briquetted at Woodward before being charged into the blast furnace; cement is used as binder. Three belt feeders take the screen products and discharge them into Stephens-Adamson weigh hoppers which discharge to railway cars. The company hauls the ore to Woodward with its own



Two Hardie-Tynes horizontal compressors supply 3000 cfm for drilling in the mine.

railway equipment. Table III shows average analysis of the ore hoisted during 1949.

Particular attention is paid to health and safety by the Woodward organization. Underground tank sprays are used on all main line haulage to keep dust under control. At the surface crushing and screening plant, a Roto-Clone dust collection system is used. There is a chief safety inspector and one inspector at each of the Woodward mines who hold monthly safety meetings for the supervisors. There are four fully trained rescue teams. A mine rescue station is maintained at the coal mines with a fully equipped mine rescue truck which is available to all of the mines. There is sufficient apparatus to equip two rescue teams. Resuscitators are kept on the rescue truck and at the lamp house of the Pyne mine. Two Joseph A.

Table III
Average Analysis of Ore Hoisted from Pyne Mine

	Pct
Silica	13.54
Alumina	4.03
Lime	16.41
Magnesia	0.51
Iron	33.63
Phos.	0.28
Manganese	0.15

Holmes safety certificates have been won from January 1947 to January 1950 without a fatality during which period 3½ million tons of ore were produced by 3¾ million man-hr of work.

From the time of the opening of the Pyne mine development, work has been kept well ahead of stoping so that there has never been a problem of finding working places from which quick tonnage can be taken. Production can be expanded or curtailed without pressure on development crews. An idea of the degree to which this is carried out can be gained from Table IV.

A monthly mine development report is prepared for the use of the supervisory staff and the management. Below is a reproduction of the form

Table IV
Development Work Progress

Started July 1942	Ft Driven	Stope Heading on Hand in Ft
1942	2,850	
1943	14,925	
1944	21,750	6,000
1945	17,633	23,706
1946	28,874	35,142
1947	25,162	44,602
1948	18,317	51,150
1949	12,795	49,154
1950 through August	12,306	49,945

used with sample figures of a few representative working places. Referring to Figs. 1 and 3, all the headings from 7 South were driven prior to 1949. Heading 701 has 977 ft of heading not available because it is now used as a main return air course. Heading 702 has thirteen stopes of 780 ft available, and 171 ft not available because it represents pillars at panel headings. Heading 703 with thirteen stopes has seven worked out, three working, and three still available. In feet; of 300 ft available at the beginning of the month, 120 ft or 2 stopes were used during the month, leaving 180 ft available when needed. One hundred sixty-eight ft are in pillars at panel headings not to be mined until ready to abandon this area.

In 704 heading all thirteen stopes are worked out and only the 162 ft of panel entry pillars remain. In 705 heading all stopes have been mined.

These reports provide a quantitative picture of the mine which when supplemented by the mine maps enables the staff to easily project production and development plans and statistics.

WOODWARD IRON CO.

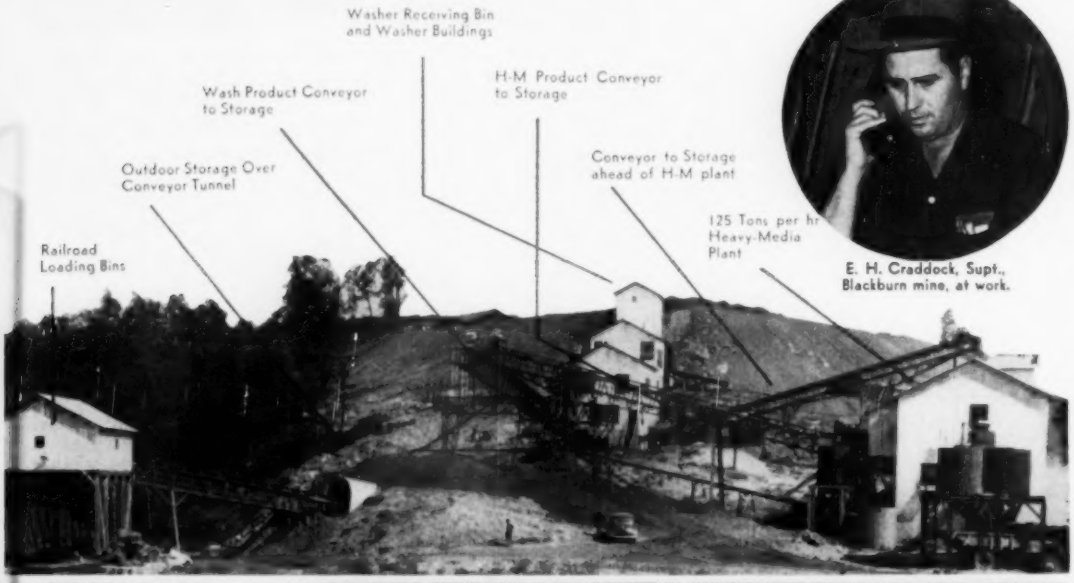
MINE DEVELOPMENT REPORT

PINE MINE

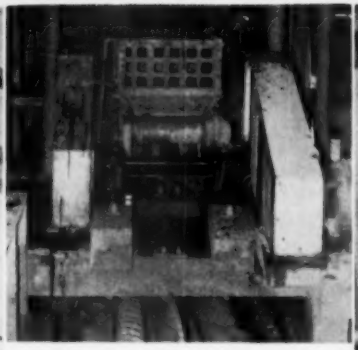
MONTH OF August 1950 ALL DISTANCES GIVEN IN FEET

NAME OF PLACE	TOTAL DRIVEN DURING YEAR OF 1949	TOTAL DRIVEN THIS YEAR TO AND INCLUDING MONTH OF July	AMOUNT DRIVEN TO MONTH OF Aug. FLUE	AMOUNT DRIVEN TO MONTH OF Aug. STOPE	TOTAL THIS YEAR TO DATE 9/1/50	NO. OF ROOMS OR STOPES TOTAL	NO. OF RMS OR STOPES WORKED OUT	NO. OF RMS OR STOPES NOW WORKING	NO. OF RMS OR STOPES STILL AVAILABLE	ROOM OR STOPE HEADINGS NOT AVAILABLE DRIVEN PREVIOUS	DRIVEN THIS MO.	DRIVEN TOTAL	AVAILABLE	USED	BALANCE AVAILABLE
703										977	0	977			
702						13	0	0	13	171	0	171	780	0	780
703						13	7	3	3	168	0	168	300	120	180
704						13	13	0	0	162	0	162			
705						14	14	0	0	0	0	0			

(1) Mining and stripping is done by 3 draglines excavating 3000 cu yd of material a day. At left, a Lima 802 is digging with a Hendrix 3-cu yd bucket. **(2)** At center, 15-ton Euclid rear-dump trucks powered by 150 hp Cummins engines make 1 mile trip between pit and washer. Truck at right carries float tailing from mill. **(3)** Some ore is stockpiled to keep washer at work when pit is shut down. Shovel (right) loads stockpiled ore for transportation to receiving bin of the washer.

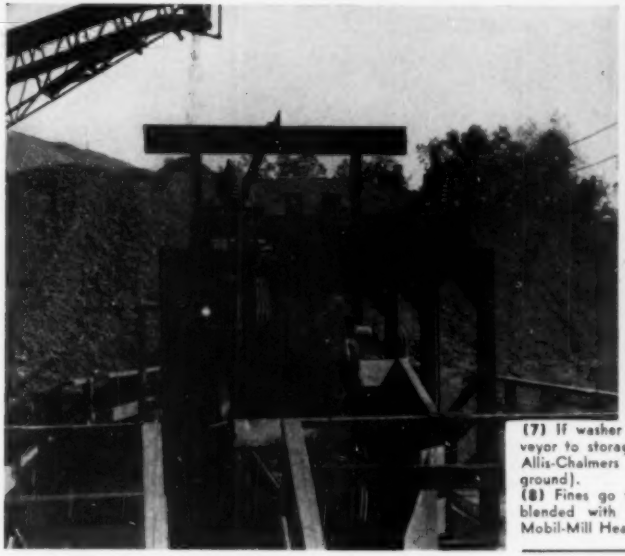

Washer Receiving Bin and Washer Buildings
H-M Product Conveyor to Storage
Wash Product Conveyor to Storage
Conveyor to Storage ahead of H-M plant
Outdoor Storage Over Conveyor Tunnel
125 Tons per hr Heavy-Media Plant
Railroad Loading Bins

E. H. Craddock, Supt., Blackburn mine, at work.



(4) Left, ore is washed at 100 psi at receiving bin of washer. Plus 3-in. material is crushed, joins—3 in. at log washer. Screening of washer product catches slabs which are returned to crusher, undersize goes to **(5)** above, second log washer in foreground, in closed circuit with McClanihan 18x20-in. stone crusher (middleground). Overflow is pumped to settling pond, and entire washer product is screened **(6)** on Allis-Chalmers 3-deck screen. Plus 1½, ½, and 1/16-in. screen products are combined in the dividing bin. If free of rock, washer product goes to railroad outdoor storage.

Heavy-Media Separation Increases Brown Ore Reserves

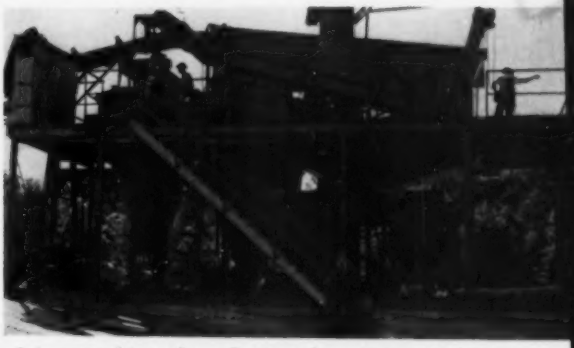


(7) If washer product needs further treatment it goes via conveyor to storage over tunnel conveyor. Tunnel conveyor goes to Allis-Chalmers 5x16-ft low-head, double-deck screen (middle-ground).
(8) Fines go to this dewatering classifier. Dewatered fines are blended with shipping product from Western Machinery Co. Mobil-Mill Heavy-Media plant.

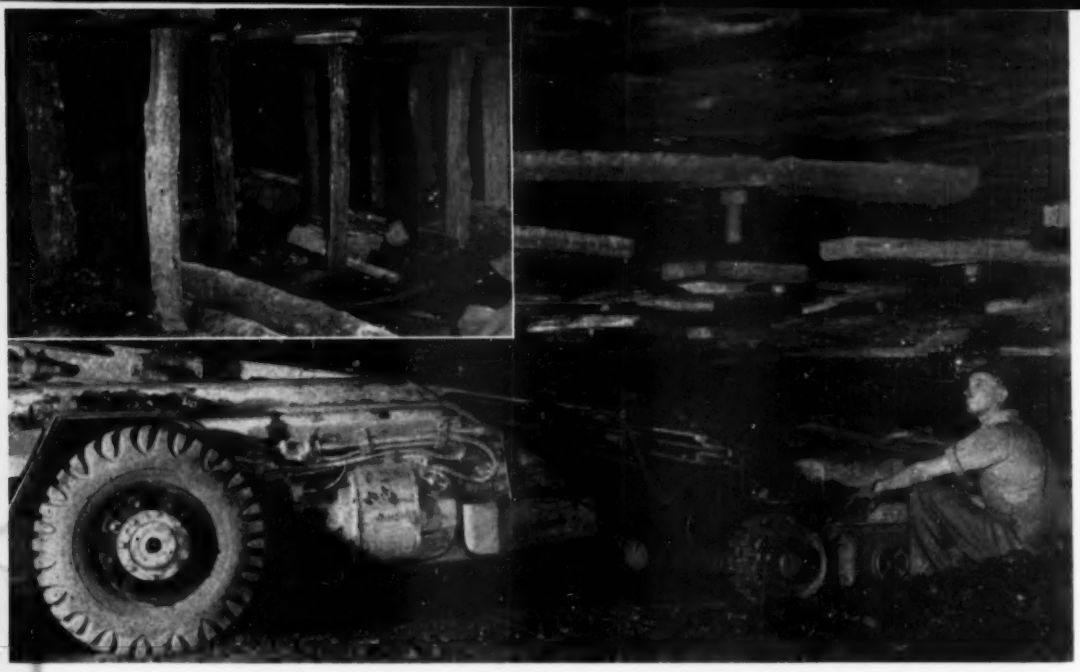
BROWN iron ore reserves of Franklin county are being proved economical by Heavy-Media separation at the Blackburn mine of the Shook & Fletcher Supply Co. about 120 miles northwest of Birmingham. The ore occurs under a cover of ferruginous chert and gravel and these materials are also disclosed as streaks in the ore body as it is mined. Previous methods of jigging and hand sorting the ore became too expensive because of rising wages but the use of Heavy-Media separation has proved economical and is making Franklin county the most promising source of additional reserves of brown ore.

Brown iron ore production in Alabama amounts to about a million tons per year. It was first used in making iron and is still sought after for mixing with red ores because it increases furnace capacity by its higher iron content and reduces coke consumption because of a lower melting point. Tuscaloosa and Bibb counties, southwest of Birmingham, and eastern parts of the State also have the brown limonite ore deposits. For a description of mining and concentrating at the Blackburn mine follow the picture story.

(9) The ½-in. material from double-deck screen is processed by Heavy-Media separation in 10-ft diam cone shown above. Ferrosilicon medium maintains 2.80 sp gr at top, 2.90 at bottom of cone. Sink product blended with fines is 47 pct Fe, dry analysis.



It is conveyed to outdoor rail storage, from whence a 250 fpm tunnel conveyor transports it to railroad bins. (10) The 125-tph Western Machinery Co. Mobil-Mill, shown here before housing, has made possible the recovery of iron from gravel previously wasted.



Greater Coal Production at Gorgas Mine

by Milton H. Fies

manager, Coal Operations, Alabama Power Co., and consulting engineer.

THE Alabama Power Co. operates its Gorgas mine in Walker County, Ala., immediately adjacent to its steam plant. The coal mine property consists of some 19,000 acres of fee and mineral ownership lands, and all of the production from the mine—approximately 2500 tons per day—is utilized at the company's adjoining steam plant. The company is presently considering the purchase of additional acreage adjoining its present holdings which acquisition will almost double its reserves. The coal is transported from the mine over a short railroad spur in the company's own equipment, either directly to one of the large plants at Gorgas or to a storage pile. The longest haul is about 1¼ miles.

The mine produces 2500 tons of washed coal per day of two shifts. The preparation of the coal, resulting in a uniform 3x0 in. size product, is obtained by passing the coal through a Bradford breaker, thence to two Jeffrey diaphragm jigs, thence over dewatering screens to a 1000-ton capacity coal bin for further dewatering and transhipment to the power plants. Flow sheet A indicates the travel of the coal from mine car to railroad bin.

The coal property has been thoroughly drilled and otherwise prospected and contains some 85 million tons of proven recoverable coal in three coal beds. These coal beds, beginning at the surface, are the Pratt, and at an interval of 40 to 70 ft below, the American, and then at an interval of 400 ft, the Mary Lee. Coal is being mined only from the American bed, but resumption of mining from the Pratt bed is planned for 1952.

The data presented relate to the operation of one Joy continuous miner, which began to produce coal regularly in March 1950, and to the application of roof bolting throughout the entire mine, which began at the same time.

The American seam at Gorgas averages about 54 in. in thickness. A typical section of the seam is shown in Fig. 1.

Fig. 2 presents a section of the mine showing the method of mining with rooms and pillars where Joy mobile loaders are employed. The rooms are 25 ft wide, driven 250 ft deep, and the pillars between rooms are 20 ft across. Since the American workings are in advance of the Pratt, no pillars are pulled. The width of the rooms compared to that of pillars is increased with the continuous miner and the percentage of recovery appreciably improved.

The larger portion of the production of the mine other than that obtained with the continuous miner is procured with mobile loading machines, Joy 12 BU, loading onto conveyors. These conveyors load either onto belts in the cross entries, or directly into Sanford-Day 6-ton drop bottom mine cars. The main line track is laid with 70 lb rail, a part of which is thermite welded, with plans for eventually welding the entire track. This track is slag ballasted, and treated ties are used. Main line locomotives are 15 tons, gathering locomotives 6 to 10 tons.

The bottom below the coal is fire clay, which is fairly soft in its consistency. Directly above the coal in varying degrees of hardness and thickness there are two draw rocks. The first draw rock, which is the most difficult to hold, is from 2 to 6 in. thick, and above, the other draw rock varies from 10 to 20 in. in thickness. In most sections of the mine only the first draw commonly falls with the coal as it is undercut and shot. Occasionally the second draw comes down, but it is only in rare instances that both draw rocks, which might total 30 in., cannot be supported. The rock above these draw rocks is a hard, sandy shale, ideal for anchoring roof bolts.

During a period of from six to eight months in 1949, roof bolting was applied to a set of three places where a Joy mobile loading unit was in operation. Then the practice was extended to some three or four additional units. The adaptability of the system to the conditions as they exist at Gorgas was definitely proved before the bolting was applied to the entire mine. Before any definite decision was reached, many tests were made in various sections of the mine, and

By Roof Bolting and Continuous Miner

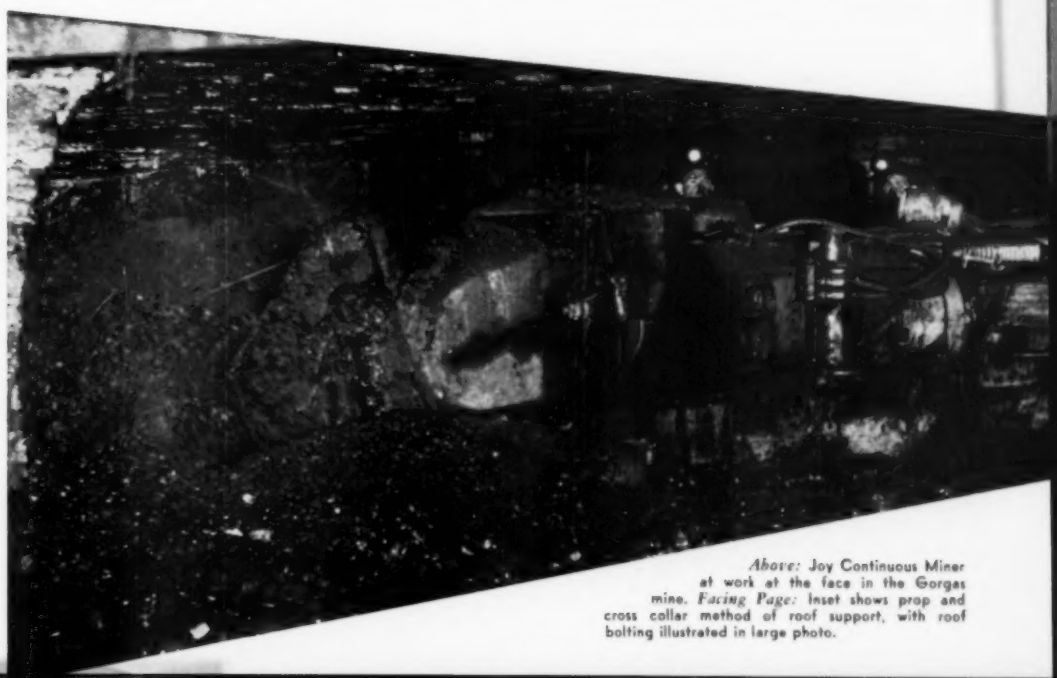
in other operations in the district. The results obtained were helpful in the final decision to adopt the system throughout the entire mine.

Experience demonstrated that one of the most important factors in securing satisfactory roof bolting was to obtain a fixed anchorage for the bolts in the roof stratum. Where the wedge-type bolt is used, the nature of the rock stratum in which the anchor is located, the design of the wedge, the method employed in slitting the bolt end, and the hardness of the bolt steel itself, are all factors affecting the holding power or strength of the anchor.

It is recommended that each individual mine using the wedge-type anchor do a certain amount of experimenting and testing in order to determine that the anchorage is sufficient to support the amount of rock required.

Tests made in pulling roof bolts at Gorgas and other mines, using a Continental Gin roof-bolt tester, with gage calibrated to register tension pull directly in tons, gave some interesting results. For example, at Gorgas a 1-in. cut thread, burned-slot bolt was originally used with a $^{3}/_{4}$x$^{3}/_{4}$x5$^{1}/_{2}$-in. wedge in a 1$^{3}/_{8}$-in. drill hole. This bolt installation provided anchorage (at 8-ton ram pressure) sufficient to break bolts in all tests. Later, in an effort to reduce the cost of the roof bolts (approximately 13,000 were installed per month at this mine) it was decided to use a 1-in. rolled-thread bolt with a sheared slot. The shank diameter of this sheared-slot bolt is approximately 29/32 in., and its cost is 12$^{1}/_{2}$¢ less than the cut thread with burned slot.

When this latter type bolt was first installed on a trial installation, a $^{3}/_{4}$x$^{3}/_{4}$x5$^{1}/_{2}$-in. burned wedge was used in a 1$^{3}/_{8}$ in. drill hole, the same as has



Above: Joy Continuous Miner at work at the face in the Gorgas mine. *Facing Page:* Inset shows prop and cross collar method of roof support, with roof bolting illustrated in large photo.

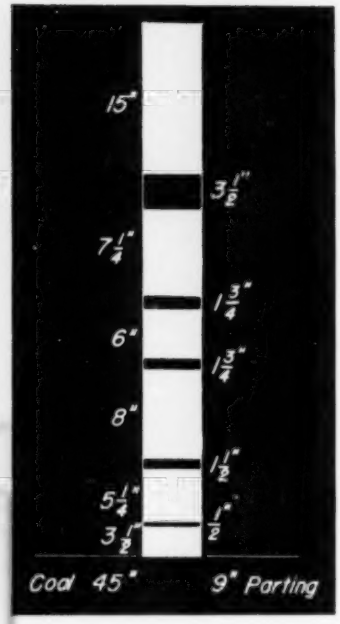


Fig. 1 Typical section of seam at Gorgas mine.

been used with the 1-in. cut thread with burned slot. These bolts however, would pull out at around 5-ton tension, irrespective of the method used in driving the bolt into the roof hole (hydraulic ram, stoper, and hand driving was tried). It was found that by anchoring them in a 1¼-in. hole and using a larger wedge, $^{5}/_{8}$x$^{3}/_{4}$x6 in. burned wedge, and 8-ton ram pressure, the bolt would break before pulling out. It was further found that by using a wedge designed so that the bolt could be rammed to the extreme depth of the hole at approximately 8 to 9 tons pressure, maximum holding was obtained.

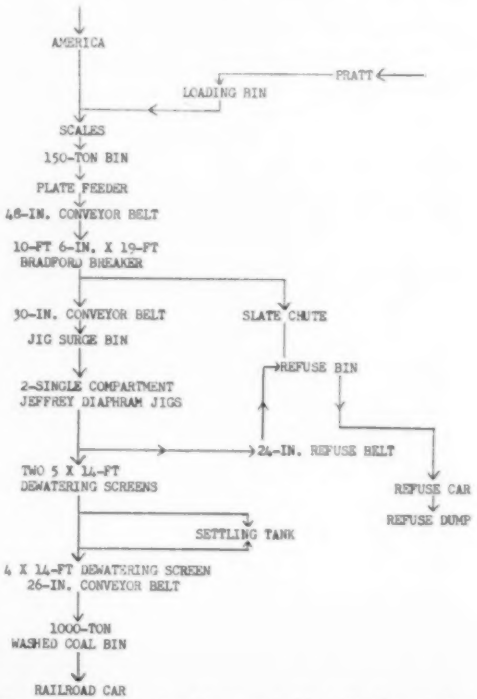
At a second mine a number of tests were made with various types of wedges, and the only wedge-type bolt that could be made to hold over 10 tons was a high grade carbon steel bolt.

In some cases better anchorage can be obtained by increasing the length of the bolt and anchoring it in more suitable stratum. Experience indicates that the expanding shell-type anchor is not affected by the variations in rock strata as much as are the wedge type; and where it is difficult to secure anchorage with the wedge-type bolt, the expanding wedge shell may offer the best solution even though the installation time and expense may be greater.

The roof bolt pull-test data presented herein, Table I, were compiled from a test made at the Gorgas mine to determine the holding strength of various type bolts under varying methods of installation—wedge design and ram pressure. The roof holes were drilled with a rotary electric drill, and the bolts installed with an hydraulic ram.

It soon became obvious that this method of roof suspension offered so many decided advantages over the older system of room props with cross collars that, after all possible checks had been made as to the safety for the men, the system was adopted. It was found, for example, that because of the added maneuverability of machines (such as the boom on the loading machine, or the cutting machines working across the face) the efficiency of loading units was increased from 20 to 25 pct. The contrast in the two systems of roof support is emphasized by the pictures heading this article. One (inset) shows the method of supporting the roof with props and cross collars, and the other by roof bolting.

It is extremely difficult to support the second draw in most of the working places in the mine by means of props and cross collars because of the consistency of the underlying fire clay. When the timbering in the mine was done by cross collars and props, the props would sink into the bottom as weight came on, and there was continuous trouble. Idle periods, at times only from Friday night to Monday morning, often resulted in reduced production on Monday due to falls in the working places, which required time to clean up before loading could begin. During longer idle periods it was always necessary to send crews into the mine to remove the falls of rock, and often to reset cross collars. Entries and air courses driven in advance of rooms required additional cleaning when the equipment was moved forward. Experience demonstrated that it was particularly troublesome for the face crews to carry props and 3x8x16-in. cross collars for timbering to the face in such medium thickness coal, and the time lost was appreciable. This work was made much less laborious with the new system where it is required to carry only the



Flowsheet A—travel of coal from car to bin at the Gorgas mine.

lighter materials, consisting of bolts, nuts, and small size T-boards.

Roof support by means of bolts has no dependence on the nature of the floor of a mine, and that gain is important where the bottom is soft.

It has been demonstrated that the ability of bolts to support the roof is greater than the strength of props and cross collars, but it has been decided to place the bolts on the same centers as the props, that is 4-ft squares. The maximum draw rock encountered had reached 30 in., and bolts were fixed at a 40 in. length, which would give an 8-in. penetration into the permanent stratum when the maximum thickness of draw rock was encountered.

While expansion bolts and other types have been tried, the bolt adopted thus far has been the split bolt with a wedge. Table I indicates the resistance of these bolts to pull, which emphasizes the supporting strength of the bolt. The factor of safety of the roof bolting, assuming that the rock to be supported is 4x4 ft by a maximum of 2½ ft thick, is much greater than that attained in the use of timbers. And since both draw rocks in the mine average less than 18 in., the factor of safety is high.

Since in the advanced workings of this mine the supporting rock is shaly sandstone, which has become so hard that it is difficult to drill, and since the draw rock in this area has not exceeded 18 in., the length of the roof bolt has been reduced to 30 in. When the bolts were first installed in rooms it was decided to use wooden T-boards against the roof just inside of the washer and nut of the bolt. These were originally 3x8 in. x 3 ft, and were used only in rooms. Experience has justified the reduction in the size of the T-boards to 3x6 in. x 2 ft. In the entries or haulageways where roof bolts are used, timber is not employed directly under the roof, but only the metal plate washers, 6x6x¼ in. thick.

Fig. 3 (next page), shows means of supporting roof where only 6x6x¼ in. plate washers are used. Note the freedom of maneuverability of the machine.

Prior to the adoption of roof bolting throughout the entire mine and during the preliminary stages of the work, it was felt that if an effective, flexible method of drilling the roof could be devised, one that would not involve such costly

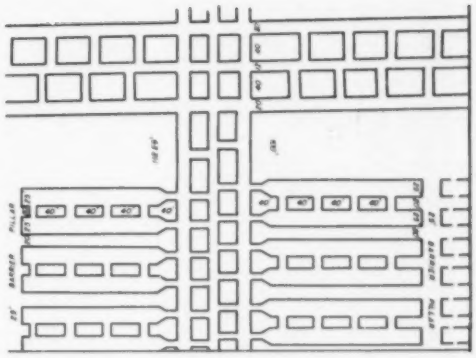


Fig. 2—A section of the mine shows room and pillar mining where Joy mobile loaders are used. Use of the Joy Continuous Miner has increased width of rooms and increased the percentage of recovery.

initial expenditure as is required for portable air compressor, jack hammer, impact wrench and piping, approval to adopt the system throughout the entire mine could be more readily obtained. Each loading unit would require the air equipment mentioned, and at Gorgas there were a maximum of nine such units. It was believed also that a rotary electric drill would release less fine rock dust in the drilling operation and that vibration caused by pneumatic stopers, which have a tendency to loosen the roof, would be eliminated by the use of the drill ram.

A light, compact, self-propelling drill rig was designed by Paul E. Glaze, assistant superintendent and chief electrician at the Gorgas operation. One of the first units installed is shown in Fig. 4. These machines have proved very effective. The number of holes per day has averaged 50, and at times the total has reached as high as 80. The men who drill the holes and insert the bolts also perform other duties.

Fig. 5 presents the new type self-propelled unit for roof drilling. At present these drill rigs are being built by the Continental Gin Co. under the trade name of "Drillram." The rotary drill is so mounted on a hydraulic cylinder that upon completion of the drilling operation this drill can be swung out of position, the roof bolt inserted in

Table I

Roof Bolt Pull Test Data Gorgas Mine—American Seam June 2, 1950

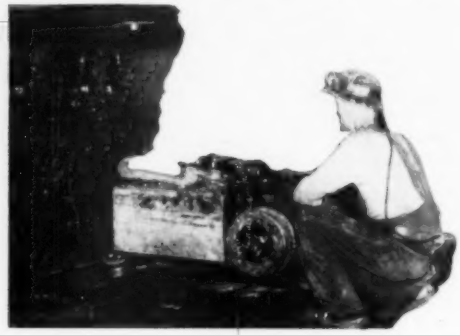
Item No.	Hole No.	3 1 in. Diam.	4 1 in. Diam.	5 1 in. Diam.	6 ⅞ in. Diam.	7 1 7/16 in. Diam.	8 1 7/16 in. Diam.	9 1 7/16 in. Diam.	10 1 7/16 in. Diam.	11 1 in. Diam.	12 1 in. Diam.	13 ⅞ in. Diam.
1	Rod Type	BS-CT (Inches)	BS-CT (Inches)	BS-CT (Inches)	ES-CT (Inches)	SS-RT (Inches)	SS-RT (Inches)	SS-RT (1) (Inches)	SS-RT (Inches)	BS-CT (1) (Inches)	BS-CT (Inches)	ES-CT (Inches)
2	Length	40	40	39 15/16	40 1/16	40 9/16	40 1/16	40	40 9/16	39 7/8	40 1/16	40 1/16
3	Thread Length	4 1/2	4 1/2	4 9/16	4 9/16	4 1/8	4 1/8	4 1/8	5 1/16	4 7/8	4 7/8	4 7/16
4	Slot Length	6	6	6 1/4	2 7/16	9 5/8	9 1/2	9 3/16	9 5/16	6 1/4	6 1/4	2 1/4
5	Bolt Diameter	1.008	1.007	0.992	0.756	0.914	0.907	0.912	0.912	0.986	0.983	0.760
6	Projection at Initial Anchorage	5 3/4	4 7/8	4 13/16	2 1/4	5 1/2	6 5/16	4 7/8	5 5/16	4 1/2	4 5/16	2 1/16
7	Projection After Push	4 3/16	3 15/16	4	——	4 1/2	4 7/16	4 1/4	3 3/4	2 5/8	3 1/4	——
8	Push Pressure	16,000 lb	12,000 lb	8,000 lb	None	12,000 lb	16,000 lb	16,000 lb	19,000 lb	12,000 lb	16,000 lb	None
9	Pull Pressure—Max.	19T	19T	9T	14T	4T	14T	11T	14T	19T	19T	14T
10	Result	Broke in thread	Broke in thread	Slipped at 9T	Broke in bottom threads	Slipped at[1] 4T	Bolt bent[2]	Slipped[3]	Bolt stretched or slipped	Broke in threads in hole	Broke in threads	Broke in lower threads
11	Diameter After Pull	0.993	0.991	0.992	0.744	——	0.880	——	0.885	Not available	0.963	0.743
12	Thread Length After Pull	5 1/4	5 9/16	4 9/16	4 13/16	——	4 13/16	——	5 1/16	5 5/16	5 1/2	5 1/8
13	Projection After Pull	6 1/16	5 1/8	——	6 1/2	——	8 1/2	——	7 11/16	5 1/2	6 1/4	6 1/2

Explanation of Symbols: BS—Burned Slot. SS—Sheared Slot. CT—Cut Thread. RT—Rolled Thread. ES—Expansion Shell. (1)—Special Notch. Holes 1 and 2 not used for pull test.

REMARKS: [1]Driven in with sledge to 3 3/8 in. proj. after slip. Broke at 14T at a point 3 1/8 in. above threads. Bolt diam. immediately above threads. [2]Roof irregularity at hole caused bolt to be pulled out of line, causing bend. [3]Driven up with sledge to 3 1/8 in. proj. no additional data available.

Fig. 3 (below)—Shows means of supporting roof when only 6x6x1/4-in. plate washers are used, giving considerable maneuverability to the machine.

Fig. 4 (left)—Self-propelling drill rig, designed by P. E. Glaze at the Gorgas operation, averages 50 roof bolt holes per day.



the hole just drilled and the hydraulic ram swung into position to set the split bolt with wedge, without relocating the machine itself.

The contribution which the roof bolting has made in the safe operation of the mine has been significant. Roof bolting, when used in conjunction with the continuous miner type of machine, where no explosives are required, will greatly minimize roof falls, the greatest hazard in coal mining. The U. S. Bureau of Mines during the years has been outstanding in its contribution to coal mining, but it may well be that in its long history no single accomplishment exceeds that of roof bolting in mines.

The cost of roof bolting at Gorgas mine is greater by approximately 10¢ per ton than that by the use of wooden supports, but this difference is small as compared to the added efficiency of the machines, the increased tonnage obtained and the increased safety of the men.

The continuous miner, purchased from the Joy Mfg. Co., of the type 3-JCM, maximum height 39 in., produced some coal during a few shifts in the month of January 1950. It was not, however, until after the mines resumed operation on March 6, following a period of idleness due to a nationwide strike, that the work of the miner became regular. Only one continuous miner is employed in the mine at Gorgas at the present time. It operates on two shifts and largely in rooms 17 ft wide, which have a 10-ft pillar between. It will be noted in this connection that this compares with the 25-ft rooms and 20-ft pillars employed in the conventional method of mining, and that the percentage of recovery is markedly increased, Fig. 6.

When the machine was first installed, an attempt was made to operate it with one operator, one helper, one car trimmer, and one shuttle car operator, using the intermediate shuttle car, operated by push button on the miner, as a surge. This is not practical in coal 52 to 54 in. thick, for the reason that the operator of the miner does not have sufficient height over the surge shuttle car to properly gage its loading. After the first few shifts a second shuttle car operator was added. The crew then consisted of the machine operator, the helper, two shuttle car operators and a car trimmer, who loaded the mine car.

Coal was mined at the face, conveyed by the shuttle car to a belt on the cross entry, and from the belt the coal was loaded into a 5½-ton mine car on the main entry. This method of operation was continued through March 1950. Time studies were made on each shift, and it was soon definitely determined that the idle period when the machine was standing was due entirely to the inability of the two shuttle cars to effectively serve it. It was then that one of the assistant superintendents of the mine, John Davis, conceived the idea of utilizing an additional mobile loader, a small 12 BU, between the continuous miner and the intermediate shuttle car. The bottom of the mine acts as the surge. The continuous miner, as it produces the coal, deposits it on the floor and the 12 BU picks it up and loads it into the intermediate shuttle car. It has been the experience at Gorgas that the miner is thus made continuous.

Table II shows the cost per ton, separated as to day and night shifts, of coal produced by the continuous miner from January 1, 1950 through August 1950 delivered in the mine car on the main haulage entry. The intermediate Joy mobile loader began operation during April 1950. It will be noted how the tons per man-day have increased with the proportionate decrease in the cost per ton. The figures indicated are raw coal, but tests reveal that the tons of washed coal per ton of raw coal with the continuous miner is 85 pct; whereas, in the conventional method of mining, using undercutting machines, explosives, and mobile loaders, the washed coal is only 72 pct of the raw coal produced on the same basis.

It has been estimated that on a washed coal basis the cost of producing coal with a continuous miner in the mine car on the main entry is about one third of the cost of producing washed coal with the conventional mobile loaders.

The crew now consists of one continuous miner operator, one helper, one Joy operator, two shuttle car operators, one car trimmer, and one foreman. The average number of tons produced since March per shift has been 297.5 of raw coal, with an average 85 pct wash coal recovery. The greatest advance in solid coal in one shift has

Fig. 5—Continental Gin Co.'s Drillram, a new type of self-propelled unit which drills holes for roof bolts, then sets the split bolt with wedge by means of a hydraulic ram.

Table II

Cost Per Ton of Coal Produced at Face with Continuous Miner, January 1, 1950 to August 31, 1950

Date	No. Men Shifts	No. Shifts	Total Tons	Tons Per Man	Amount	Cost Per Ton
January	56	11	1714	29.6	$ 805.52	0.47
February	Mine idle this month due to strike.					
March	250	36	7390	29.6	3,557.62	0.48
April	301	40	11,352	37.7	4,313.61	0.38
May	335	46	12,629	37.7	4,869.98	0.39
June	304	44	14,089	46.3	4,396.44	0.31
July	216	32	9413	43.6	3,147.65	0.33
August	323	45	14,190	43.8	4,614.55	0.32
Total	1787	254	70,777	39.6[1]	$25,705.37	0.36[2]

[1]Deduct 15 pct to obtain tons washed coal per man-day.
[2]Divide by 0.85 to obtain cost per ton washed coal.

Table III

Sample Screen Analysis of Coal Taken From Continuous Miner

Date	+6 in. Pct	-6 in. +3 in. Pct	-3 in. +1½ in. Pct	-1½ in. +¾ in. Pct	-¾ in. +½ in. Pct	-½ in. +⅜ in. Pct	-⅜ in. -8M Pct	-8M -0 Pct	Total Pct
6-12-50	0	4.2	7.3	19.75	10.2	17.88	17.6	23.1	100.03
6-13-50	3.98	6.28	10.26	19.29	9.80	15.93	15.64	18.3	99.5
6-14-50	6.71	6.54	10.91	19.80	9.89	13.93	14.76	17.4	99.94
6-15-50	3.6	4.3	10.8	20.9	11.3	14.4	16.4	18.3	100.
6-16-50	4.2	5.5	11.7	20.7	10.1	14.1	15.0	18.6	99.9
Average	4.6	5.4	10.2	20.09	10.26	15.25	15.9	19.1	100.8

Table IV

Sample Screen Analysis of Coal From Joy Loaders

Date	-6 in. Pct	-6 in. +3 in. Pct	-3 in. +1½ in. Pct	-1½ in. +⅜ in. Pct	-⅜ in. +½ in. Pct	-½ in. +⅜ in. Pct	-⅜ in. -8M Pct	-8M -0 in. Pct	Total Pct
6-12-50	4.4	7.7	11.8	21.6	10.1	12.7	15.6	16.0	99.9
6-13-50	11.2	11.5	17.8	23.9	7.0	9.8	9.6	9.1	99.9
6-14-50	8.28	10.51	15.44	21.48	7.83	10.96	12.37	13.1	100.0
6-15-50	3.3	8.6	18.0	24.1	8.8	11.1	12.0	14.0	99.9
6-16-50	5.3	12.6	21.1	25.5	6.7	9.1	9.8	9.8	99.9
Average	6.5	10.2	16.8	23.3	8.1	10.7	11.9	12.4	99.9

been 156 ft, and the largest tonnage per shift has been 443 tons of raw coal.

The savings in explosives cost amounted to about 8¢ per ton.

The noticeable improvement in the condition of the roof in the places where the continuous miner has operated has been most marked. Ribs of pillars are not shattered and accordingly are capable of supporting more weight. Likewise, the hazard of falling rib coal is greatly minimized. Draw rock which formerly came down as the coal was shot, now stands up in places after the supports are withdrawn for as long as three months. The 17-ft rooms are timbered with cross collars with 6-ft centers supported by light jacks. These rooms are timbered by the operator and helper as the places advance. After each room is driven to its boundary, which is on an average of every two shifts, the cross collars and jacks are withdrawn to the nearest crosscut and are there available as the adjoining inby rooms intersect these crosscuts. In the entries and air course the roof is supported by bolts.

The use of the continuous miner at Gorgas has convincingly demonstrated that: (1) This method of winning coal from the face is dependable; (2) the use of the machine greatly reduces the hazard of mining (a) by the elimination of explosives, (b) by reducing roof exposure, (c) by reducing man-hours of exposure per ton of coal mined, (d) by concentration of rock dust, (e) by removing the worker from the face (the operator of the machine performs his task at a minimum of 10 ft from the face), (f) by eliminating the shattering of ribs of pillars and thus avoiding falls of pillar coal; (3) the cost of producing coal is greatly reduced.

The matter of the size of the coal produced is of no moment at a mine like Gorgas where the coal is produced for steam purposes only, and this likewise would be true at mines producing metallurgical coal. Sizing tests shown in Tables III and IV over a period of a week indicate the difference between sizes of the coal with a conventional mobile loader and undercutting machine as compared to the continuous miner. No noticeable effect has been observed in the quality of the washed coal due to the use of this machine, producing about ¼ of the total production.

The results thus obtained in the use of this one continuous miner warranted the recent purchase by the Alabama Power Co. of two additional miners. If such results continue, the mine should be completely equipped with these machines.

The writer feels a deep sense of obligation to the following members of the mining staff of Alabama Power Co. for their criticism and suggestions in the preparation of this paper: Harry M. Johnstone, manager of Gorgas mine, J. W. Shealy, senior engineer, W. O. Hodgkin, mining engineer, and E. W. Booker, superintendent.

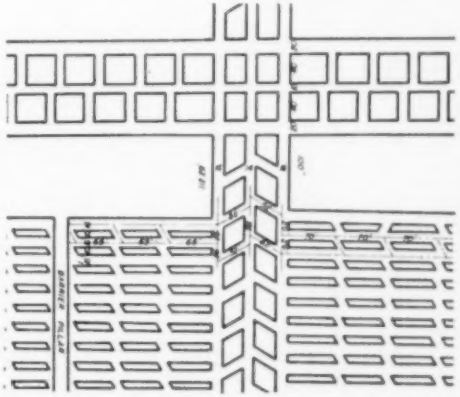


Fig. 6—A markedly increased percentage of recovery has resulted from use of the Joy Continuous Miner, with rooms now 17 ft wide with a 10 ft pillar. This contrasts with the 25-ft rooms with 20-ft pillars used in conventional mining.

ABC Typifies Trend to Mechanized Mining and Coal Preparation

by H. J. HAGER

assistant general superintendent of mines

Alabama By-Products Corp., Birmingham, Ala.

THE Alabama By-Products Corp. is an entirely independent Alabama enterprise. Organization of the company stemmed from the consolidation of three important properties in the Birmingham District and a by-product coke plant at Tarrant, Ala. In 1924 ABC was merged with the Pratt Consolidated Coal Co. This merger greatly extended the company's holdings of valuable coal lands in Walker and Jefferson Counties.

The major operation of the company is the production of uniform foundry coke. There are now in operation 174 Koppers ovens; however, to protect existing capacity and insure continuous supplies to vital defense industries throughout the country, the company has presently under construction an additional battery of 29 ovens. Of almost equal importance is its varied mining operations. Seven mines are now producing metallurgical, steam, and domestic coal from two major seams; namely, the Black Creek and Mary Lee. The Bradford and Thermal mines in the Black Creek seam produce coal almost exclusively for the coke plant. Two mines on the Mary Lee seam, Barney and Samoset, are strictly commercial operations producing both steam and domestic sizes. The other three mines on the Mary Lee seam, Colta, Labuco, and Praco, comprise a group delivering coal to a central washing plant located at Praco. From this plant is produced a high grade metallurgical coal and various sizes for the general steam trade.

Barney Mine—A Typical Commercial Operation

Barney mine, located in the eastern part of Walker County, is about 45 miles northwest of Birmingham. The mine is served by the Southern Railroad and also has facilities for water shipments on the Mulberry fork of the Black Warrior River.

Two methods of mechanical loading are employed at Barney; namely, Joy loading onto chain conveyors, and scraper loading with three-drum

Praco preparation plant is rated at 350 tons of washed coal per hr. It is designed to blend, prepare, and load highest quality metallurgical and steam coal from 100 pct inside Praco coal or any combination of outside and Praco coal.

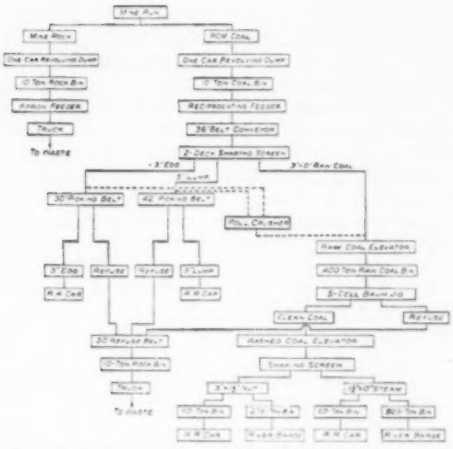


Fig. 1—Barney preparation plant hand picks lump and egg coal screened from 3-in. x 0 which is jigged and sized to 3 x 1½-in. nut and 1½-in. x 0 steam coal.

Sullivan hoists on a circular V-wall. Coal is loaded into 2½-ton mine cars for transportation in 24-car trips to a rotary dump outside the mine. The mine is operated two shifts per day, producing 1000 tons.

The Mary Lee seam in the Barney area ranges in thickness up to 35 in. The top is a tough slate and the bottom is hard fire clay. The seam is generally flat lying, however, occasional "swags" are encountered which require grading for haulage. The seam is clear but relatively high in inherent ash.

The mine is developed on a panel system. Panel entries are on 600-ft centers and cross entries are on 1500-ft centers. Panels are driven in sets of four, 30-ft wide on 60-ft centers. No. 1 entry is brushed in bottom for roadway clearance. Cross entries are driven in sets of two, 30-ft wide on 60-ft centers, for room-and-pillar mining and in sets of four for v-wall mining. In the latter case, the two center places are brushed for haulage and the two outside places are used for wall up-sets. All headings are driven with 12 BU Joy loaders and chain conveyors. Roadway brushing is done by hand on a contract basis. Practically all rock is gobbed.

The Joy loader work is on a conventional room-and-pillar system. Rooms are driven 35-ft wide on 55-ft centers. Rooms are worked in pairs, advancing on the right side and retreating on the left side of the entry. A short wall machine with 7½-ft cutter bar is used in each room.

In areas where caving of the surface is not objectionable and good top prevails, v-wall mining is employed. Walls are either worked on the right as the double-entry advances, or right and left on retreat. Each v-wall setup mines a block of coal 250x500 ft and produces approximately 6000 tons. The three-drum hoist is located on the off side of the track from the center of the wall. The wall face is normally started from the right rib of the air course to the left, the wall being advanced in a semicircle to the right. The average wall will work about one-half out, that is, to a line at right angles to the entry, before taking weight. When this occurs the sheaves are reset and the remainder of the wall is worked by starting on the left rib of the air course to the right and advancing the wall to the left.

A wall crew consists of four men. One combination man runs the short wall mine machine and operates the hoist. The other three face men jointly perform the other work of drilling, shooting, timbering, etc.

Fig. 1 gives a flow sheet of the Barney preparation plant. From a receiving hopper under the rotary dump, raw coal is fed to a double-deck shaking screen by means of a 36-in. belt conveyor. The 3x6 in. and + 6-in. sizes are hand picked and loaded as domestic coal. The 3 in. x 0 from the shaking screen is handled by two bucket elevators to a 400-ton jig feed bin. Also, any portion of the + 3-in. coal can be passed through a double-roll crusher for mixing with the normal 3 in. x 0.

From the raw coal bin, coal is fed by apron feeder to a 5-cell McNally-Pittsburg Baum-type wash box. Washed coal travels by bucket elevator to a shaking dewatering and sizing screen. Various sizes of nut and washed slack are loaded through 50-ton bins into railroad cars or placed in a 1000-ton washed coal storage bin for barge loading.

Early in 1920 when the company started producing coke for the general foundry trade, it was found that no one Alabama coal made a satisfactory coke. Therefore, after exhaustive laboratory and full size oven tests, a blend of Mary Lee and Black coals was found to give the most desirable coking results. As low volatile, sulphur and ash contents are essential for maximum blending effects, a limited amount of low-volatile Pocahontas coal is added to the mix.

A typical analysis of the three coals used in the oven blends is given in Tables I, II, and III. These coals are mixed in various proportions depending on the type coke desired.

TABLE I—Washed Black Creek Coal

Proximate Analysis	Pct
Volatile	33.12
Fixed carbon	64.22
Ash	2.66
Sulphur, total	0.64
Ash softening temp., °F	2650
Btu	15,332

TABLE II—Washed Mary Lee Coal

Proximate Analysis	Pct
Volatile	29.95
Fixed carbon	59.55
Ash	10.50
Sulphur, total	0.88
Ash softening temp., °F	2750
Btu	13,951

TABLE III—Low-Volatile Pocahontas

Proximate Analysis	Pct
Volatile	16.96
Fixed carbon	76.68
Ash	6.36
Sulphur	0.63

Labuco, formerly operated by hand loading onto chain conveyors, has been converted to mobile loading in the past two years. One main panel entry 3000 ft long was developed to the property line. This panel entry is equipped with a 36-in. mother belt. Haulage in cross entries is by a 30-in. conveyor belt. Two cross entries are worked to produce 2000 tons raw coal on two shifts

The Mary Lee seam at Labuco presents many mining problems. The seam varies in height up to 13 ft and contains three major rock partings. Fig. 2 shows a typical cross-section of the seam in this area. The seam is undulating, with grades up to 12 pct being encountered. The roof is a friable mixed slate and sheet sand rock which requires close timbering.

Cross entries in sets of three are developed off the main panel on 700-ft centers. Entries are 20 ft wide on 40-ft centers. The 30-in. belt is located in the center entry. Entries are advanced with 8 BU Joy loaders and chain conveyors. Rooms, 30 ft wide on 60-ft centers, are worked on the advance on the right side of entry and on retreat on the left side. Three rooms are worked simultaneously. Each room is equipped with chain conveyor discharging onto entry belt. A short wall mining machine with 7½-ft cutter bar is maintained in each room.

Due to the nature of the immediate overburden, roof control requires close attention. Formerly 6x8 in. x 16 ft cross bars were set on 5-ft centers with a minimum of four posts to each cross bar. Recently roof bolting has been employed. However, this has not been found to be entirely successful. The present method of timbering rooms is by the use of 1-in. x 6-ft roof pins on 5-ft centers followed by a row of posts on 6-ft centers.

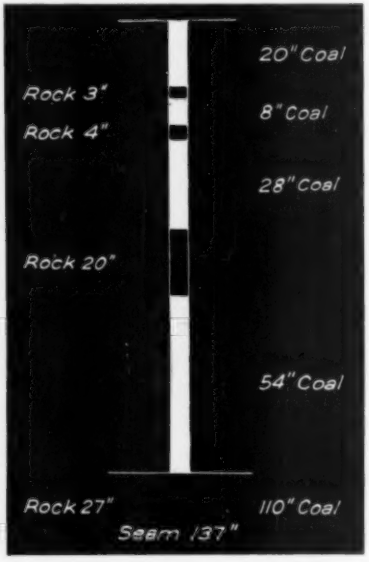


Fig. 2—Typical section of the Mary Lee seam in the Labuco mine shows three major rock partings, more than 9 ft of coal.

The 36-in. panel belt discharges into 4-ton drop-bottom mine cars. Trips of 12 cars are handled on inside slope 1500 ft long by electric hoist, thence by 15-ton locomotive 3000 ft to tipple.

Coal is discharged from the mine cars into a 75-ton receiving hopper. It is then fed by apron feeder to a Bradford Breaker. The 2 in. x 0 coal travels by means of a 24-in. belt direct to railroad cars for transfer to the Praco plant. Refuse is handled from the breaker by 30-in. belt to a 50-ton bin. Disposal is by dump truck.

Praco is a slope operation with the slope extending from the outcrop 8000 ft to the basin of the seam. Two-ton capacity mine cars are hoisted to the surface in trips of 24 cars. Two 800 hp electric hoists pull the trips to the surface. The upper half of the slope is double track, the lower half being single track.

The Mary Lee seam at Praco ranges up to 10-ft in thickness. That part of the seam in excess of 8-ft contains rock partings similar to Labuco. The smaller size coal contains two rock partings of 4- to 6-in. thickness located near the top. The seam pitches about 1½ pct in a southeasterly direction.

Praco Mine

The mine is developed on a panel system. Panel entries, in sets of four, are driven 12 ft wide and 7 ft high. Panels are 3200x3400 ft. Cross entries on 390-ft centers are driven in pairs 30 ft wide on 60-ft centers. The cross entries are developed to the next panel before rooms are started, as full retreat mining is employed in each panel. In the low coal, rock is brushed in the bottom to provide track clearance 9 ft wide and 6 ft high.

Rooms are started from the inby end on the first entry in a panel. Each successive entry is worked in like manner so as to develop and maintain a 45° retreat or break line. Rooms are driven 30 ft wide on 45 ft centers. The worked out portion of the entry caves as mining progresses outward. Two rooms are worked on each entry. A maximum of five entries are worked simultaneously on a panel. Mining in both rooms and entries is by 12 BU Joy loaders and chain conveyors. Each room conveyor discharges into mine car on entry. A short wall mine machine is provided for each working place.

Praco is gaseous and is ventilated on a split system. Ventilation is induced by two axial-flow fans exhausting in excess of 400,000 cfm. A unique phase of the ventilating system is that each panel is completely sealed as it is worked out.

Having in mind the demands of the coke plant, the Praco washery was designed to blend, prepare and load the highest quality metallurgical and steam coal.

The preparation plant, using principally Link-Belt equipment, was put into operation in June 1944. Facilities are available to assure a continuous supply of blended raw feed to the plant. The blending arrangement can handle 100 pct inside Praco or any combination of Praco with outside coal delivered either by truck or railroad car. The rated capacity of the plant is 350 tons per hr on a washed coal basis.

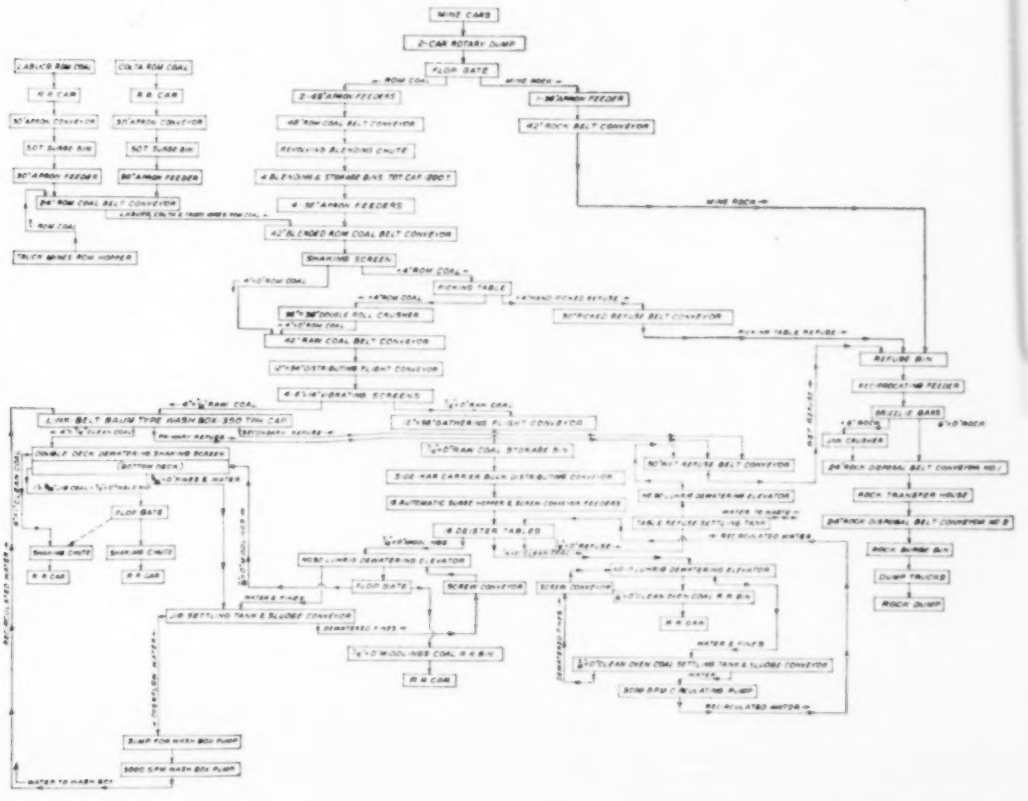
Fig. 3 shows a flow sheet of the Praco plant. The loaded trips are pulled by auxiliary hoist and dumped in a two-car rotary dump without uncoupling the cars. Power operated fly gates beneath the dump divert the contents of the cars into either small coal storage or small rock storage bins.

The coal is moved out of the hoppers under the dump by two 48-in. apron feeders to 48-in. belt conveyor which delivers the coal to the raw coal silos at the rate of 900 tons per hr. A revolving chute is provided at the discharge of the 48-in. belt so that coal can be uniformly fed to all four circular silo bins or confined to one bin for later blending. Run-of-mine coal is drawn from the silos by means of four variable speed apron feeders onto a 42-in. conveyor belt for transfer to the primary screening station.

Also fed to the 42-in. belt is coal from Colta, Labuco, and truck mines. Coal from railroad cars is unloaded, with the aid of a Robins car shakeout, into two track receiving hoppers. Coal is moved out of the hoppers by apron conveyor into two 50-ton surge bins; thence by apron feeder to 36-in. belt which discharges into the 42-in. belt under the silos. This arrangement of blending silos and surge bins for outside coals permits accurate mixing of coals from various sources and assures a raw coal feed to the washing units of uniform sieve and float-and-sink analysis.

Mixed run-of-mine coal is delivered to a combination shaking screen and picking table. Due to the relatively small sizes of the coal, a single deck equipped with lipped plates containing 4-in. openings is used. As the major portion of the

Fig. 3—Praco flowsheet indicates flexibility in blending and washing by these features: two-car rotary dump; 1000-ton, 4-compartment blending bin with 4 roll feeders; primary sizing and crushing to 4 in.; secondary screening to separate jig and table feed; 16 tables; Baum jig; sizing and dewatering washed coal; dewatering and mixing metallurgical coal. Heavy-density retreatment of secondary jig refuse is being studied.



material above 4-in. is rock, the coal is picked from the rock. The hand-picked coal is passed through a single roll crusher set at 4-in. opening.

The 4-in. x 0 raw coal is delivered by 42-in. belt to a scraper type distributing conveyor where it is discharged onto four 6x14-ft Robins Gyrex screens. The vibrators are equipped with ½-in. square mesh screens. The 4x4½-in. material is fed by scraper conveyor to a Baum-type wash box. The ½ in. x 0 coal is delivered by scraper conveyor to a 100-ton table-feed storage bin.

Float-and-Sink Data on 4 in. x 0 Praco Raw Coal

Specific Gravity	Weight Pct	Cumulative Wt. Pct	Ash Pct	Cumulative Ash Pct
Float 1.35	50.6	50.6	7.6	7.6
1.35 - 1.40	8.8	59.4	15.3	8.8
1.40 - 1.45	4.3	63.7	19.1	9.5
1.45 - 1.50	2.7	66.4	24.2	10.1
1.50 - 1.60	5.5	71.9	24.0	11.1
Sink 1.60	28.1	100.0	80.3	30.6

Float-and-Sink Data on ½ in. x 0 Praco Raw Coal

Specific Gravity	Weight Pct	Cumulative Wt. Pct	Ash Pct	Cumulative Ash Pct
Float 1.35	70.0	70.0	7.2	7.2
1.35 - 1.40	10.1	80.1	15.5	8.2
1.40 - 1.45	4.2	84.3	18.4	8.7
1.45 - 1.50	2.2	86.5	24.5	9.2
1.50 - 1.60	1.9	88.4	33.4	9.7
Sink 1.60	11.6	100.0	71.3	16.8

The Mary Lee seam is noted for its difficult washing characteristics, the coarser sizes being closely laminated with boney coal. It will be noted from the float-and-sink data on the 3 in. x 0 and the ½ in. x 0 sizes that the fine sizes will produce a lower ash washed coal. With this factor in mind, the Praco plant was originally designed so that all metallurgical coal would be prepared on 16 Deister-Overstrom tables, with the feed consisting of the normal 5/16 in. x 0 increment of the 4 in. x 0 raw coal. The washed coal ash in this product was 9 pct. The 4 x 5/16 in. size was washed on a 6-cell Baum Link-Belt jig. This coal was marketed as steam fuel at 13 pct ash. As the demand for coking coal increased with a concurrent decrease in demand for steam grades, it was found necessary to increase the production of oven coal. This was initially accomplished by changing the table feed size from 5/16 in. x 0 to ½ in. x 0. Due to further change in market conditions, metallurgical coal composed of mixed table coal and jig coal in the following sizes: 1½ in. x 0 and 4 in. x 0 is now being furnished. The ash content of the mixed coal is from 10.0 to 10.5 pct.

The table feed is 15 tons per hr of ½ in. x 0 coal at 16.8 pct ash. The washed coal, at 11 tons per hr, contains 9.60 pct ash. About 8 pct middlings are produced. Due to the sizing action of the tables, a large portion of the middling product is fine size coal. Therefore, the middling is screened at 1/16 in. The 1/16 in. x 0 size, amounting to 50 pct of the middling product, is sufficiently low in ash to be mixed into the washed coal.

To avoid segregation, the table plant is equipped with a unique feeding arrangement. A smaller hopper with a capacity of 1500 lb is installed above each table and is equipped with a tapered spiral feeder. The hoppers are fed by means of a bulk carrier conveyor (a new design by Link-Belt Co.) which receives coal from the 100-ton bin. This conveyor consists of a series of overlapping buckets, equipped with rollers, attached to a single strand chain traveling in a horizontal plane. Each bucket is provided with an undercut gate which is tripped to open over each hopper.

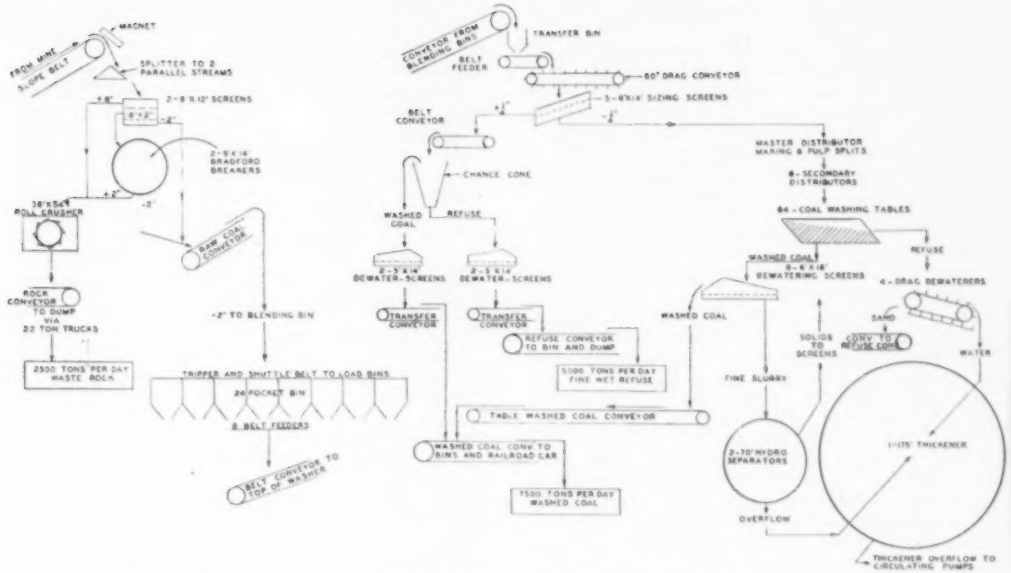
The 4 in. x 0 raw coal is flumed to the 2-compartment wash box. As various portions of the jig coal are now being used in the coking coal mix, the jig is operated at the lowest possible gravity in order to obtain a minimum ash content in the washed coal. Due to the fact that the Mary Lee coal contains a high percentage of near gravity material, the above washing practice produces a secondary refuse containing a relatively large amount of float coal and middling material. Exhaustive tests by the research department at the coke plant indicated that some form of heavy-media separation would give the most effective results in retreating the secondary refuse. Consequently, a Western Machinery Co. No. 3 Mobil-Mill with drum separator is now being installed. The secondary refuse will be crushed to 1-in., the 1x¼ in. being treated in the heavy-media unit, and the ¼ in. x 0 added to the table plant feed. It is estimated that 20 pct of the secondary refuse amounting to approximately 100 tons per day will be recovered at a sufficiently low ash to be added to the coking coal mix.

Coal from the jig passes to a Parrish double-deck sizing and dewatering screen. The products + and — 1-in. can be mixed or loaded separately into railroad cars.

The clean coal from the table plant is flumed to an elevator sump, thence by Luhrig elevator to two 6x14-ft Robins dewatering screens. The dewatered table coal can be discharged into 50-ton railroad storage bin or transferred by belt conveyor to be mixed with the jig products at the discharge of the shaking screen. The water from the table coal elevator sump and the jig coal dewatering screen is carried to the back end of two concrete settling tanks. These tanks, containing about 2200 cu ft each of settling volume, are equipped with scraper conveyors. The drags discharge the settled fines into the elevators above the water line in the sump.

All washer refuse, together with mine rock, is discharged into a 50-ton refuse bin. After crushing to — 6-in., this material is transferred by two 24-in. belt conveyors on 388-ft centers to a small bin on top of a hill. From this bin refuse is placed on the disposal dump by 15-ton diesel truck.

The need for improved preparation of metallurgical coal is constant. The Alabama By-Products Corp., in its efforts to produce quality products in both coal and coke, is ever aware of new developments in both mining practices and preparation facilities.



New Concord washer of TCI is designed to produce washed coal with any desired ash content, maximum recovery at predetermined ash content, maintain predetermined quality within close limits, and to operate without interruption for breakdowns.

New Preparation Plant Features Modern Design and Equipment

by William S. Springer,

preparation engineer, coal mines engineering department,
Tennessee Coal, Iron & Railroad Co.

A NEW preparation plant has been put in operation to treat coal from the recently opened Concord mine, located about 15 miles west of Birmingham, Ala., by the Tennessee Coal, Iron & Railroad Co., a U. S. Steel subsidiary. The mine produces metallurgical grade coking coal from the American seam for use as coke in the company's blast furnaces at the Fairfield and Enseley works. The capacity of the operation is 1000 tons of run-of-mine per hr, approximately half of which is delivered to the by-product coke plant at Fairfield after beneficiation to metallurgical grade. The remaining portion of the mine run is rock and refuse removed during preparation. No boiler coal or other fuels are produced by this mine.

Preparation of the coal is done both underground and on the surface. The underground primary crushing station removes some tramp iron, and screens and crushes the oversize, most of which is rock, to protect the main slope conveyor. The surface plant consists of a crushing and screening plant, large blending bins, a two-process washing plant, a water clarification and

reclamation system, a control laboratory, a repair shop, and accessory facilities.

The preparation facilities were designed and built for these basic requirements: (1) Equipment and facilities capable of adjustment to produce a metallurgical washed coal at ash contents to meet quality specification changes that might be made from time to time; (2) Maximum coal recovery when operating to produce any predetermined ash content; (3) Control features for holding variations in product quality within close limits to ensure production of a uniform ash washed coal; (4) Spare equipment, service facilities and bin storage capacity so most maintenance work can be performed during operating shifts without interrupting mine operations; (5) High employee morale by facilities to maintain a clean plant and pleasant surroundings.

Run-of-mine coal, unloaded from automatically spotted 7-ton, 8-wheel mine cars in a 2-car full revolving rotary dump, falls into a hopper below the dump. Material is fed from this hopper through two parallel lines of equipment, consisting of 60-in. apron feeders, 60-in magnetic pulley conveyors, 5x10-ft scalping screens and 36x54-in. single-roll crushers. The —7-in. undersize through the non-blinding tapered-opening grizzly deck of the screens and the —8-in. crushed oversize drop to a double surge bin. From this bin two 36-in. apron feeders load the —8-in. run-of-mine coal on a short center 42-in. buffer conveyor which brings the flow of material up to the speed of the main slope conveyor to which it discharges. The distance from the service crane above the rotary dump to the bottom of the sump below the buffer conveyor is 106 ft.

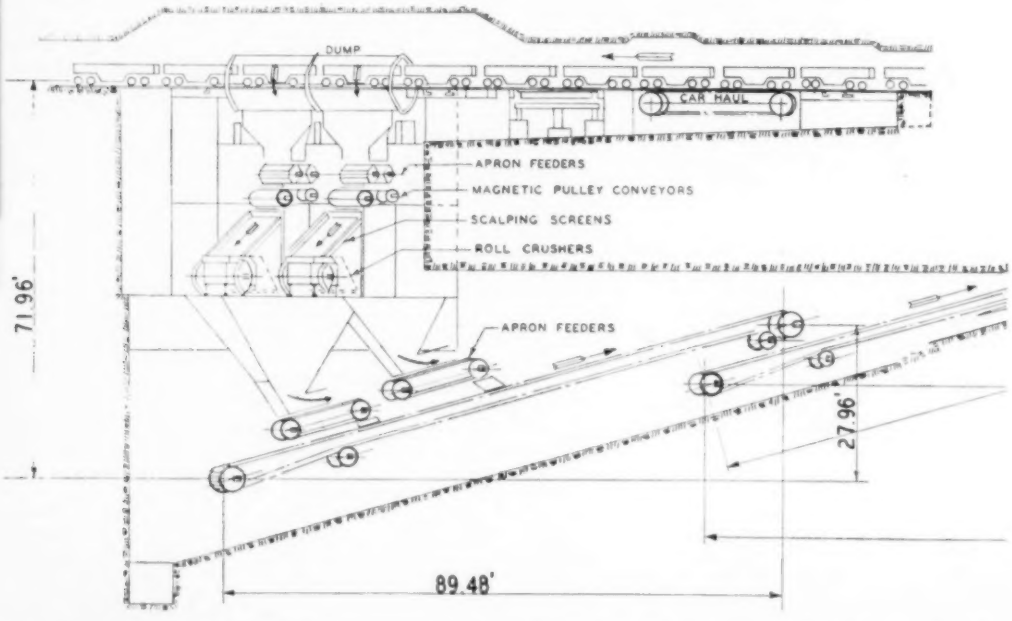
At the time of installation the Concord slope conveyor represented the largest belt installation ever made insofar as hp and lift on one belt were concerned. It can be considered a pioneer installation because it developed belt stresses and produced problems previously unsolved. Many major changes have been made since the initial installation, but in no case to date have these caused more than nominal interference with production.

The 42-in. steel-cord slope belt is about 4900 ft long on 2450-ft centers. This conveyor lifts the coal a vertical distance of 730 ft at a peak rate of 1000 tons per hr and a belt speed of 575 fpm. The conveyor is driven by three 3 hp wound-rotor motors on two drive pulleys.

The American Seam coal contains two rock partings having an average thickness of 8 in. for the top band and 24 in. for the bottom band. The coal is friable and this characteristic, along with the heavy underground shooting required to break up the rock partings, results in a large production of fine sizes, about two-thirds of the —2-in. prepared feed for washing being in the —¼ in. sizes.

Due to the extreme friability of the coal, revolving breakers are provided which easily remove the +2-in. rock amounting to about 20 pct of the mine production.

A cell-type blending bin is provided to produce a feed for the washing plant uniform in the amount of rock impurities and the size distribution. Blending is essential since large variations occur in the mine-run coal, due to part of the production being obtained from areas where only the seam above the large rock parting is extracted, while the remainder comes from areas where full seam extraction is practiced.



The underground primary crushing station removes some tramp iron, and screens and crushes the oversize, most of which is rock, to protect the main slope conveyor which, at the time of installation, was the largest belt in horsepower and lift on one belt.

For ultimate recovery a two-process cleaning plant is provided. The 2x¼-in. size is separated at 1.50 to 1.60 sp gr, depending on product requirements, in a cone which uses a sand-water medium. The ¼ in.x0 size is separated at 1.70 to 1.80 sp gr, depending on product requirements, on concentrating tables.

Dewatering is effected on horizontal vibrating screens for all products, including the thickened fine sizes from both the hydroseparators and the thickener. The water circuit which circulates 5000 to 7000 gpm, depending on plant feed tonnage, is clarified by two 70-ft hydroseparators and a 175-ft thickener.

Plant Flowsheet

Run-of-mine coal discharges from the mine slope conveyor at the top of the crusher building where a rectangular suspended magnet removes scrap iron that was not recovered by the underground magnetic pulleys. This flow passes over a splitter car where it is divided equally to feed two parallel lines of equipment. Each line is capable of handling near mine capacity in emergencies and a breakdown of any equipment in either line automatically shifts the splitter car to direct the total flow to the other line.

Each line consists of a 6x12-ft double-deck vibrating screen and a 9x14-ft revolving breaker. A grizzly type top deck on the screen is provided with openings which taper from 4½ in. at the feed end to 8½ in. at the discharge end to prevent wedging and blinding. The oversize is all rock and is by-passed to a 36x54-in. single-roll crusher. The oversize from the bottom deck, on 2-in. sq openings, discharges to revolving breaker having 2-in. round perforations in its shell. Plus 2-in. rock is removed from the breaker through a peripheral discharge which goes to the same crusher as the scalped rock for reduction to —3 in. The rock is crushed as a dump fire control measure. The crusher product, amounting to approximately 20 pct of the mine production, is carried on a 36-in. belt conveyor to a 150-ton rock bin for disposal on the dump by 22-ton semi-trailer trucks. The —2-in. undersize material from both the screen and the breaker combine on a 48-in. belt conveyor for transfer to the top of the blending bin.

Coal is received at the blending bin by a 60-in. tripper belt which feeds a 60-in. shuttle belt. The traveling tripper-shuttle unit fills the bin pockets for blending and is designed for the shuttle belt to reverse its direction of travel under load without stopping the flow of coal. Twenty-four pockets are provided in the bin, each divided into four compartments, which are arranged in three lines of eight pockets each. Eight 30-in. variable speed feeder belts receive coal from the air-operated bin pocket draw-off gates. Each belt serves the gates from three pockets which are controlled to prevent more than one of the three gates being opened at the same time. In practice, one line of eight pockets is completely filled before coal is removed from the line, the pocket compartments being filled in order from bottom to top. When one line is filled, a second line is started and coal removal is changed to withdraw from the filled line. When necessary, however, coal withdrawal can be from pockets that are not in line. About two thirds of the 600-ton bin capacity is available for surge capacity in cases of emergency breakdowns to permit continuation of mine hoisting.

Blended coal is delivered to a 25-ton surge bin at the top of the cleaning plant by a 54-in. belt conveyor which connects it with the blending bin where the conveyor is fed by the eight belt feeders. A 72-in. variable speed belt feeder removes coal from the surge bin and discharges to a 60-in. scraper conveyor. This conveyor distributes the coal to five 6x16-ft double-deck vibrating screens, each installed at a 22° slope, for wet screening at ¼-in. The top deck screens have ¾-in. sq openings serving as a protective deck for the bottom screens which have 5/16-in. sq openings. Four rows of water sprays are provided for each deck. Spray water and the feed push water are regulated to give a 1½ : 1 water to solids ratio in the undersize pulp.

The +¼-in. oversize material from both decks, amounting to 35 pct of the plant feed, recombines and collects on a 42-in. belt conveyor and is carried to a 12-ft diam cone for cleaning in a sand-water medium. The cone separates the feed into washed coal and refuse in the proportion of one part washed coal to three parts refuse. Each product is removed from the medium and dewatered on one of two 5x14-ft horizontal vibrating screens. The medium returns to the circuit and each product discharges to a 30-in. transfer belt conveyor.

The undersize pulp from the raw coal sizing screens, amounting to 65 pct of the plant feed, flows to a primary distributor of company design. This unit divides the feed pulp into from two to eight parts, depending on plant feed tonnage, with all parts being quantitative and qualitative equals. Each part feeds a secondary distributor which further divides the pulp to feed each concentrating table.

In principle the primary distributor consists of a revolving top cylinder having seven outlets which discharge below to any one of three stationary compartmented pans. Each stationary pan has a different number of compartments, namely four, five, and six. Discharge from the bottom pan is directed to an arrangement of eight flumes through swivel discharge spouts, one outlet being provided for each compartment. A fourth bottom pan having seven compartments is also provided which is held stationary to effect seven splits or is revolved to make eight splits. The predetermined number of splits can be directed to the flumes feeding whichever secondary distributors are to be operated.

The secondary distributors are designed similar

to the primary unit, except only a seven-eight way split can be made. A change in the number of splits for both primary and secondary distributors can be made without stopping the flow of pulp. This feature permits any one of eight tables in a table section to be removed from the circuit for maintenance or one or more complete table sections to be shut down.

Eight table sections, each containing eight tables, or a total of 64 tables, separate the feed into washed coal and refuse in the proportion of six parts washed coal to one part refuse. The table refuse flows to four drag dewatering tanks, each tank serving two table sections, or 16 tables. The settled refuse solids are dewatered on the inclined end of the tank and discharged by the drag conveyor to a 30-in. collecting belt conveyor. The water overflowing the tanks flows to a 175-ft diam thickener for further clarification.

The table washed coal is dewatered on eight 6x16-ft horizontal vibrating screens equipped with ½-mm opening stainless steel cloth. Each screen serves a table section of eight tables. Twenty five sq ft of stationary wedge wire screen with ½-mm openings is located ahead of each vibrating screen to remove part of the water.

The water and solids passing through the screens combine in a common flume which splits to feed two 70-ft diam hydroseparators. These units make a nominal size separation at about 270 mesh. Sizes below 270 mesh are contained in the water which overflows the tanks and go to the 175-ft thickener for further clarification. Sizes above 270 mesh settle and thicken, and are raked to the center where they are withdrawn into a sump at 45 pct solids.

One constant speed pump and two variable speed pumps for each tank handle the underflow pulp. The variable speed units are provided with a hydraulic coupling which is automatically controlled by a liquid-level control instrument to regulate the pump speeds between set limits. The pumps discharge to a distribution tank which sends the pulp back to a feed box located over each of the table washed coal dewatering screens. The pulp is put on the screen as a second layer about one-third of the length of the screen from the feed end where all free water has passed through the screen and the bed of solids can serve as a filter medium to trap the fine solids which previously had passed through the screen.

Solids in the thickener underflow are withdrawn at 50 pct solids by two diaphragm pumps and discharged to a centrifugal-pump sump. Either of two variable speed pumps, similar to those provided for the hydroseparators, pumps the slurry to either a tailing pond or to the table washed coal dewatering screens where it is admitted as a third layer. The ash content of the slurry and the washed coal specifications govern the direction of flow.

The dewatered table washed coal discharges from the eight screens to a 48-in. collecting belt conveyor. A 54-in belt conveyor receives the total washed coal from both the cone and the tables and delivers it at 18 to 20 pct moisture to a 500-ton capacity bin for loading in railroad cars on two tracks. A 36-in. belt conveyor receives the total refuse from both the cone and tables and discharges to a 450-ton capacity bin for disposal on the dump by 22-ton semitrailer trucks.

Clarified water overflows the thickener at about 1 pct solids, collects in a sump and is returned to the plant circuit by pumping in two stages, three pumps being available for each stage. Plant make-up water is obtained through use of fresh water for pump glands, spray nozzles and wash-down hoses. The thickener is provided with a high outside wall and serves as a collecting reservoir for all the water in the plant circuit when the plant is shut down, or when a power failure occurs.

Plant control is maintained by having each feed and product belt conveyor equipped with both a weightometer to obtain accurate tonnage figures and automatic sampling equipment which meets ASTM standards. A complete analytical laboratory furnishes control data that are necessary to ensure production of a uniform product of predetermined ash content.

The provision for adequate bin storage capacity and the design feature of having spare equipment and pipe lines installed for all units, except several of the main conveyors and the cone separating vessel, permit all normal maintenance work to be done on operating shifts. No equipment breakdowns of any anticipated duration can interfere with mining and hoisting operations more than momentarily.

Every unit of equipment in the plant, where its operation is necessary to receive the product from a preceding unit, is provided with a governor and/or an electrical interlock. This feature requires starting of equipment in proper sequence, and stoppage of any unit for any reason automatically stops all preceding units and prevents a plant choke up. Other electrical controls, such as bin level indicators and limit and pressure switches are also tied into the operating sequence which will shut off the feed to the plant and/or sound a warning. Auto-stop-test switches are provided for each drive to permit operation of the unit, out of sequence, for test purposes.

All motors are totally enclosed and the electrical equipment is water tight which permits washing down of all working places. The switchgear for the plant is housed separately from the preparation building.

Operation of the plant is controlled from a central panel which is also provided with warning and trouble lights for the various equipment items. The warning lights indicate a particular unit of equipment is not operating normally and corrective action must be taken to prevent a plant stoppage. The trouble lights indicate a particular unit of equipment has shut down and the plant feed has been automatically stopped.

Research and Educational Facilities at the University of Alabama

by James R. Cudworth Dean, College of Engineering, University of Alabama

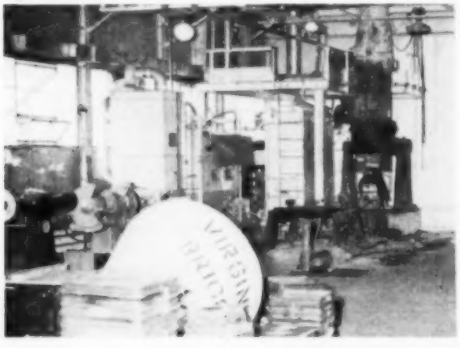
ENGINEERING has been taught at the University of Alabama for over a hundred years. The School of Mining and Metallurgy was authorized by the Board of Trustees in 1887 and four students were graduated in 1888. In 1919 the legislature directed that all mineral industry instruction in the state should be centralized at the University and designated the department of mining of the College of Engineering as the Alabama School of Mines and made financial provisions for carrying on both instructional and research work for the benefit of the mining industry of the state and for cooperation with the U. S. Bureau of Mines, which had established its Southern Experiment Station on the campus.

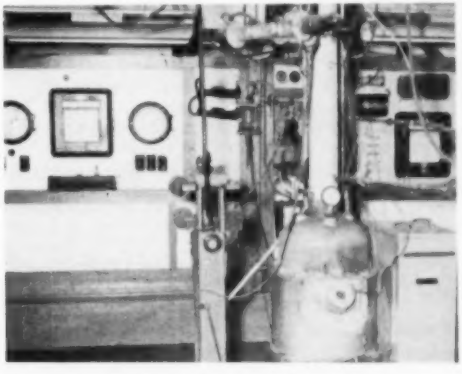
The University is ideally located for carrying on educational training for the Birmingham district. It is sixty miles from Birmingham and considerably nearer to some of the large coal and iron mines. Excellent roads make it possible to carry students to various new installations for observation of new developments. Field work for mine surveying courses can be carried on in the mines and surface geology can be studied throughout the year.

At the present time the University offers undergraduate and graduate courses in metallurgical, petroleum, and mining engineering for training for the mineral industries of the state and country. These curricula are accredited by the Engineers' Council for Professional Development.

In the School of Mines the undergraduate degree of bachelor of science in mining engineering is granted with options in the fields of general, mining, mining geology, and petroleum and natural gas. The laboratories are well equipped for mineral dressing, coal preparation, fuel testing and processing, and petroleum and natural gas studies. The modern equipment for the mining courses consists of fans, compressors, pneumatic drills, mine safety devices, and mine surveying instruments. The School of Mines maintains shops for the repair and building of equipment for research and instructional purposes.

The department of metallurgical engineering offers a curriculum leading to the degree of bachelor of science in metallurgical engineering. The laboratory facilities are ample for both ferrous and nonferrous courses. Due to the importance of the iron and steel production in Alabama, there is an unusual demand for work in this field. In 1950 the University became the southeastern unit in the Foundry Educational Foundation plan and a modern complete foundry laboratory was established with the assistance of the industry. The foundry fellowships are open not only to metallurgical engineers, but also to





Left, foundry in the metallurgical engineering department, and above, the stoker laboratory in the University's mining department. At top of page is Houser Hall, new engineering building.

mechanical, industrial, and other engineers. They take certain foundry courses and do summer work in foundries.

In the College of Arts and Sciences the department of geology offers courses for the training of geologists. In the School of Chemistry, metallurgy and ceramics courses are available on both the undergraduate and graduate level for training students in ceramics, and many of the men graduating in this field will find employment with the refractory brick and other clay product operations in the Birmingham district.

Evening courses of the first two years in engineering are also offered at the University of Alabama Extension Centers at Birmingham and Gadsden. This allows a student to continue with his employment and go to school at night for part of his college work. There are also graduate courses offered from time to time at these centers for advanced work in professional engineering.

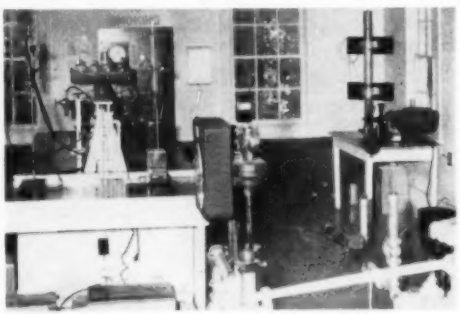
The University has accredited curricula in aeronautical, chemical, civil, mechanical, electrical, and industrial engineering and graduate work in these fields is also offered. In addition to the educational phase of its work, the University has an interest in two organized research bureaus, the Engineering Experiment Station and the State Mine Experiment Station, both of which were organized to be of service to the industries of Alabama and the Southeast. The facilities of these bureaus are personnel and equipment, and both of them are well equipped to aid the industries. Special research facilities are available in certain mineral fields such as mineral dressing, fuel utilization, metallurgy, and foundry practice. Excellent cooperation has always existed between the mining companies of Birmingham and the University.

There is need, however, to attract more men into the mineral industry engineering branches. The educational group needs the active support of the mining operators in publicizing this professional field in the high schools and in furnishing fellowships for those qualified students who need financial assistance.

The mineral industry should be cognizant that, in general, the enrollment is decreasing in the engineering schools of the country and that this will result in a scarcity of young men for replacements. This is particularly true of the mineral industry branches of engineering which have not drawn their rightful proportion of students for several years. This method of undergraduate fellowships financed by the companies has been quite successful in attracting students, particularly from the mining regions in other sections of the United States. A plan of utilizing mineral engineering students in the summer with jobs of increasing difficulty and compensation from the freshman to the senior level with a fellowship to cover tuition costs during each academic year has been successful in various mining sections.

In the difficult years to come it is increasingly important that the educational institutions work closely with the mining companies so that the needs of the mineral industries for technical services will be met.

The petroleum laboratory in the University's mining department.



About This Issue

IN presenting this Birmingham Mining Issue the cooperation of several companies and the work of a great many people was needed. To the companies and individuals listed below we give our sincerest appreciation and thanks for their splendid cooperation and work. We pay tribute to those whose accomplishments are recorded here and also to the other companies and individuals whose work has made the district great.

Alabama By-Products Corporation
- W. C. Chase, general superintendent of mines
- H. J. Hager, assistant general superintendent of mines

Alabama Power Company
- M. H. Fies, manager, coal operations

Shook & Fletcher Supply Company
- A. M. Shook, III; general manager of mines
- Edmond Craddock, superintendent of mines

Tennessee Coal, Iron & Railroad Company
- R. E. Kirk, manager of raw materials
- E. D. LeMay, director of public relations
- J. C. Gray, assistant manager of raw materials
- Leland Johnson, assistant chief engineer of ore mines and quarries
- William S. Springer, preparation engineer, coal mines engineering department

University of Alabama
- J. R. Cudworth, dean, College of Engineering
- E. C. Wright, head, department of metallurgical engineering

U. S. Bureau of Mines
- J. R. Thoenen, superintendent, Southern Experiment Station
- B. H. Clemmons, acting chief, metallurgical branch, minerals technology division

Woodward Iron Company
- B. C. Colcord, president
- J. H. Woodward, 2nd; secretary
- Hewitt Smith, vice-president in charge of mining
- J. W. Hager, chief engineer

Southeast Section, AIME
- E. H. Rose, secretary

Aspects of Structure and Mineralization Used as Guides in the Development of the Picher Field

by Joseph P. Lyden

In the Picher field, structure made openings for the circulation of the mineralizing solutions by flexing, shearing, and fracturing the sedimentary beds. This structure is used with the spatial and genetic relationship of the ore minerals, sphalerite and galena, and the gangue minerals, dolomite and jasperoid, in prospecting for ore bodies.

THE Picher Mining Field, fig. 1, which lies between Baxter Springs, Kansas, and Commerce, Okla., is the most intensely mineralized and the largest zinc-lead ore producing area in the Tri-State District of Missouri, Kansas, and Oklahoma. It has a vast lateral extent of underground workings that is unequaled for the observation of the geology of zinc-lead ore bodies in sedimentary horizons; and thousands of churn drill holes in the unmined areas supplement the underground observations with useful data for the interpretation of the geology.

This paper is confined to the ore deposits of the Picher mining field. The reader is referred to the bibliography for the general geology of the Tri-State district which has been ably covered by many geologists.

The writer was co-author of several publications on the Tri-State district.[1-4] This paper is necessarily a repetition of parts of those publications but contains a few changes and additions on the deposition of the minerals. The parts of this paper concerning mineralization maps and pipe slump-structures are not covered in the above publications.

Geology

The geology of the Picher mining field is similar to that of the entire Tri-State district because the sedimentary horizons that are found here are found throughout the district, and the ore bodies in the respective horizons have similar characteristics. The beds that contain the ore bodies in the Picher field are illustrated and described in fig. 2.

The five unconformities found in the mines of the Picher field are indicated on the geologic section, fig. 2. The unconformity shown at the base of Meramec (*B* bed) is relatively unimportant since *B* bed was mostly eroded from the area of the Picher field before the Chester beds were deposited. *B* bed has been observed in the cuttings from only a few churn drill holes.

The Pennsylvanian (Cherokee shale) formation is the top horizon found in the Picher mining field. It has been eroded from the southeastern part of the field, but because of the regional dip of the sedimentary beds to the west, it attains thicknesses of 100 ft to 200 ft along the west edge of the field. It is 200 ft to 300 ft thick in the Miami trough. It was deposited on the Boone limestone surface, which contained relatively few shallow erosion channels and very few depressions. Subsequent to the deposition of the shale, solution of limestone from underlying limestone horizons by circulating solutions caused the overlying beds, including the Cherokee shale, to slump into the larger openings formed by the solution and removal of the limestone.

Chert

The origin of the chert in the Tri-State district has been a controversial subject among geologists. Tarr[5] advocated a syngenetic origin for the chert, postulating that it formed from colloidal silica deposited in the Mississippian seas. Leith, Dake and Leighton[4] argued in favor of silicification of erosion

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AIME New York Meeting, February 1950.

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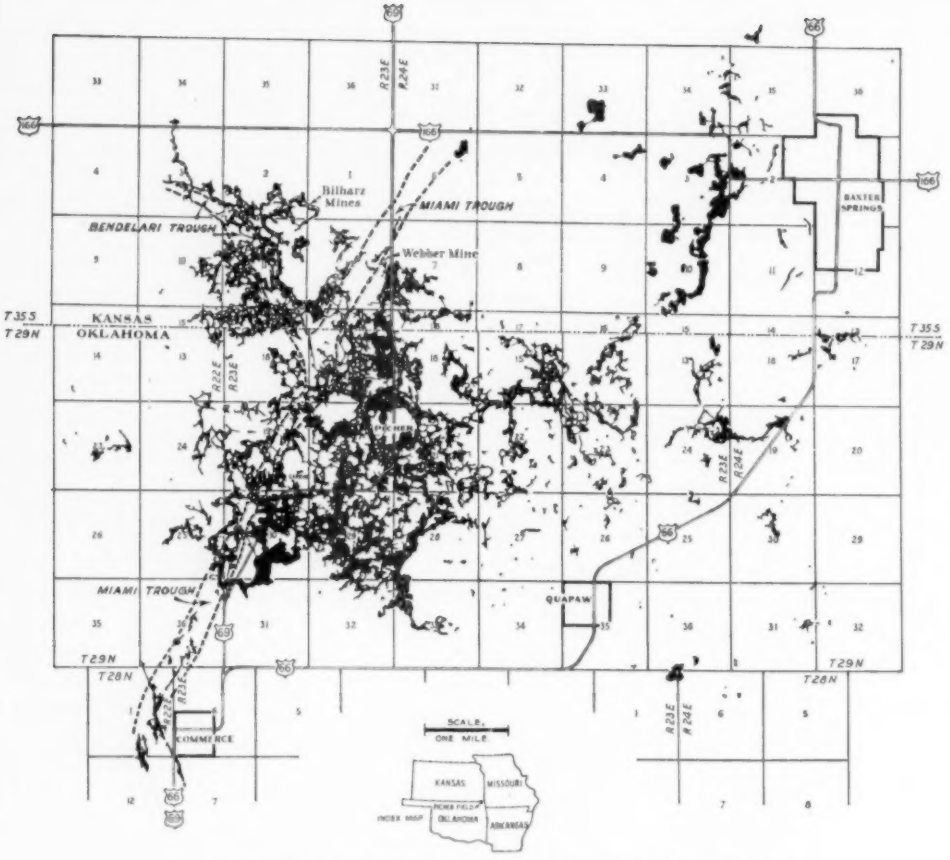


Fig. 1—Map showing underground workings in Picher Field.

surfaces, postulating the solution by meteoric waters of silica from weathered areas and redeposition in the limestone on and near the surface. Fowler, Agar, Gregory and the writer[6] advocated, and Giles[5] favored the theory that the chert is epigenetic, that it replaced the limestone, and is hydrothermal in origin.

Chert in varying amounts, fig. 2, replaced limestone, forming nodules in the more massive limestone beds, *C*, *E*, *J*, *K*, *M*, and *R*, replaced alternate limestone strata in the thin bedded limestone beds *G-H*, *O*, and *Q*, and more or less completely replaced limestone beds 15 ft to 35 ft thick, such as *D*, *F*, *L*, *N*, and *P* beds. In areas where deformation was intense, chertification was also intense and often all the beds were completely chertified in zones 100 ft to 150 ft thick. Chertification[6] took place during Mississippian time before the Chester limestone, which contains no chert, had been deposited.

Two types of chert occur, cotton rock chert and dense to glassy chert.

Cotton rock chert is a calcareous chert that permeated and partly replaced the limestone with cryptocrystalline silica. The ratio of silica to limestone varies considerably.

Tripoli, a light porous mass of silica that easily crushes to a very fine powder, is a residual product of cotton rock beds that we exposed to weathering. The limestone and silica in these beds were originally about equal in amount. The limestone was removed by solution, leaving the insoluble silica.

Dense and glassy chert, which characterizes complete silification of the limestone, is found in and close to the centers of disturbance, as are the ore bodies; whereas, cotton rock, which is characteristic of partial chertification, is more common in the respective beds in places remote from the centers of disturbance.

Ore Horizons

Ore occurs in all the beds shown below the Cherokee shale horizon in fig. 2. The limestone beds were the favorable ore horizons. The limestone was replaced by the ore minerals, sphalerite and galena, and by the gangue minerals, jasperoid and gray dolomite.

The chert beds, *D*, *F*, *L*, and *P*, fig. 2, were not favorable ore horizons because they consisted of dense massive chert which resisted replacement by the sulphide and gangue minerals. However, in the mineralized areas along the shear trends, the chert horizons were intensely sheared and fractured, mak-

ing openings in which the ore minerals were deposited in sufficient quantity to make low grade ore. There is a thin 12 in. to 18 in. limestone horizon in *D* bed that occasionally is mineralized with enough sphalerite and galena to make ore.

Shale is found in structural and solution openings in the ore horizons. This shale has been referred to as residual and transported or introduced shale. The residual shale resulted largely from the solution of shaly limestone in *J* bed. The transported or introduced shale was carried into structural and solution openings by circulating water. Fowler and the writer[1] and McKnight[9] postulate the source for much of the transported or introduced shale as the overlying Cherokee formation and believe that the shale was carried down from the base of the Cherokee through openings, many of which are found in the slump areas. Much of this residual and transported shale was replaced by the sulphide and gangue minerals. The Cherokee shale horizon, however, contains very little mineralization because it was but little penetrated by the mineralizing solutions.

Structure

The structure of the Tri-State field has been a controversial subject among geologists. The reader is referred to the discussions by Leith and Dake[1] and the article by Sales[10] for their opinions.

Structure is of the utmost importance because it was through the openings, caused by deformation of the beds, that the circulating mineralizing solutions reached the limestone horizons in which the ore bodies were deposited.

The three periods of deformation with which the ore bodies are associated are the following:

1. Pre-Warsaw Deformation: This period of deformation is relatively unimportant in the field as a whole.

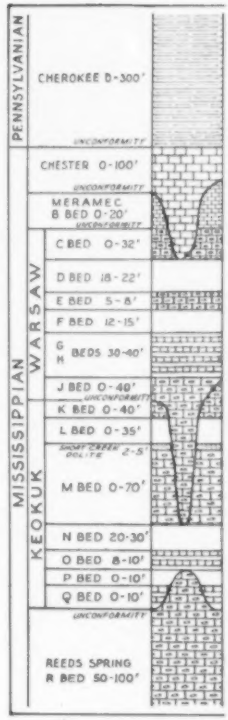
2. Post-Warsaw-Pre-Pennsylvanian Deformation: Deformation during this period produced most of the regional structures with which the ore bodies are associated. The anticlinal structures shown in fig. 3, the pipe slump-structures shown in figs. 4 and 5, the Bendelari trough shown in fig. 1, and early movement in the Miami trough belong to this period.

3. Post-Pennsylvanian Deformation: Deformation which had started in Mississippian time and continued past Pennsylvanian time produced the "Miami Trough" or "Miami Fault" as it is commonly known. This fault, fig. 1, contains along its trend a series of elongated graben fault blocks, 500 ft to 1200 ft wide, and one half to three fourths of a mile long, that dropped vertically 100 ft to 250 ft. Ore bodies have been mined within and along the edges of this trough for a distance of eight miles.

Structure Maps

Fig. 3 is a regional structure map that shows the structure on the top of *L* bed in the Bilharz Mining Company's leases. This area was selected as a type example because it shows unusually well the relations between the ore bodies and the anticlinal flexures.

Regional structure is shown by sea-level datum contours, at 5-ft intervals on the top or bottom of a bed that is below the slump horizons, and on which the most reliable information is available. The top of *L* bed, fig. 3, the top of *N* bed, fig. 5, and the top of *O* bed are the horizons often used for making structure maps.



DESCRIPTIONS OF HORIZONS

Cherokee — Dark Shale. Few sheets and lenses of sandstone including the Blue Jacket sandstone bed 25 ft thick which is 100 ft above bottom of horizon.

Chester—Limestone, partly oolitic. Contains thin shale and sandstone beds.

Meramec — Gray dense fine grained limestone.

C Bed — White and blue chert nodules in gray and light brown limestone.

D Bed — White "cotton rock" chert and dense white and light gray chert.

E Bed — Gray chert nodules in gray and light brown limestone.

F Bed—Light gray "cotton rock" chert and dense white and light gray chert.

G and H Beds — Thin bedded, gray, brown and blue chert bands alternating with brown limestone bands.

J Bed — Dark gray glauconite speckled chert in dark gray glauconite speckled limestone. Dark shaly limestone at base where J bed occupies depressions.

K Bed — White and light gray chert nodules in gray and pale brown limestone.

L Bed—White "cotton rock" chert and dense light gray chert.

M Bed — White and light gray chert nodules in gray and pale brown limestone.

N Bed—Gray "cotton rock" chert and glassy gray and light gray chert.

O Bed—Thin bedded. Gray chert bands alternating with brown limestone bands.

P Bed—Light gray "cotton rock" chert and glassy gray chert.

Q Bed—Thin bedded. Gray chert bands alternating with brown limestone bands.

Reeds Spring—Dark gray to black irregular chert nodules in dark gray and brown limestone.

Fig. 2—Geologic section with descriptions of sedimentary horizons in the Picher field.

Structure maps are used in preliminary drilling campaigns to determine the locations and trends of structures and thereby help in making locations for new prospect drill holes. They are used in subsequent development work to determine the favorable parts of all the structures and to make sure they have been prospected in all favorable horizons.

Regional Deformation

Regional deformation produced two types of structures, characterized first, by flexing of the beds, fig. 3; and second, by strong fault zones, such as the Miami trough, fig. 1. Flexing of the beds caused shearing across them and also caused movement on the bedding planes and unconformities.

Shearing across the beds is most prominent along a fold at the points of sharp flexing. The strong shearing in the flexures is usually at steep angles of 75° to 90° to the bedding, cuts across one or more beds, and is often continuous a mile or more in one direction. Ore bodies following the shearing are referred to as "runs," are long and narrow, vary from 5 ft in height in weak shearing to 100 ft or more in strong shearing, and average approximately 20 ft.

Movement on the bedding planes and unconformities opened them to the mineralizing solutions. In the horizons above *N* bed, movement on unconformities and bedding planes was mostly confined to areas within the shear zones, whereas in the horizons below *N* bed, such as *O* bed and the un-

Fig. 3—Structure contour map on top of "L" bed.
Bilharz Mining Co. leases.
Eagle-Picher Mining and Smelting Co., Geological Department.

conformity at the top of the "Reeds Spring" formation, movement was on broad flat flexures where there was little relief of the stresses by shearing across the beds. The ore bodies in *O* bed, which is known as the "sheet ground" horizon, are characteristic of this type of deformation. They average about 10 ft high and often occupy areas 100 acres or more in lateral extent.

Anticlinal, monoclinal, and synclinal structures contain ore bodies along the strong shear zones. Anticlinal structures are the most important structures in which the ore bodies are found. Most of the monoclinal structures are really parts of large anticlinal structures and can be largely included with them. Synclinal structures, while important in many parts of the field, are often barren and much less mineralized than the anticlinal structures.

In places where a mineralized syncline is flanked by mineralized anticlines, it is common to find *M* bed mineralized only in the syncline and the upper beds *G–H* and *K* mineralized only in the anticlines.

Shale Maps

Shale maps are made with 5-ft contour intervals on the bottom of the Cherokee shale horizon. They outline the post-Cherokee-age slump areas which formed because of settling and caving of the beds into solution made openings in underlying limestone horizons. Shale maps were used by engineers and geologists in the early development of the Picher field when it was observed that ore bodies often occurred around the outer part of the slumps, or "shale sinks" as they were locally known. It is a simple matter to contour the bottom of the Cherokee shale because it is easily identified in the churn drill cuttings and its depth is recorded in the many

drill hole logs. Shale maps are also useful in showing whether the structures in the Mississippian horizons formed before or after the Cherokee shale was deposited.

Slump Structures

Slump structures are secondary structures found throughout the mining field on the trends of strong regional structures. They resulted from the caving of beds into large solution-made openings in underlying limestone horizons.

The slumps formed before, during, and after the period of sulphide mineralization, and those that are mineralized often contain ore bodies from the base of the slump zone to its top. Usually the most favorable part of a slump area for sulphide mineralization is its outer edge, where favorable openings were produced when the slump block parted en masse from the surrounding rock.

The graben fault blocks in the Miami trough are slump structures, but are only partly the result of solution of underlying limestone beds. They are largely the result of regional forces that separated the fault walls and allowed long narrow blocks to drop within the fault zone.

Pipe-Slumps and Circular Shearing

Figs. 4 and 5 illustrate two pipe-slumps that occur in the Eagle-Picher Mining and Smelting Company's Webber mine. They are cylindrical in shape, are approximately 100 ft in diam, and are 300 ft or more in height. They parted from the surrounding rock on vertical-circular shearing and dropped en masse approximately 30 ft. As shown by the contours in fig. 5, they formed near the centers of basins in a synclinal flexure. Ore bodies occur in *M* bed around the pipes. The north pipe is not mineralized in *M* bed and the Short Creek oolite limestone at the top of *M* bed is exposed a few feet above the floor of the mine workings. The south pipe is partly mineralized in *M* bed and contained ore in the outer rim. Drilling has not been deep enough within and around these pipe-slumps to determine their depth and the horizon beneath them in which, presumably, openings formed to allow them to slump. The writer believes that the weight of the pipes caused the basins to form over solution-made openings, and, as the basins formed, tension forces were set up around the areas of the pipes. The resolution of the forces, the weight of the pipe, a compressive force, and the tension forces set up in the area around it, caused each pipe to part on circular shearing from the surrounding rock and slump into a solution-made opening.

Pipe-slumps are a type of structure that can seldom be observed, especially in vertical section, and for this reason there is but little reference to them in the literature. These structures are important ore-bearing structures, not only in the Picher field but throughout the Tri-State district. They formed in both pre-Pennsylvanian and post-Pennsylvanian time. The pipe-slumps of pre-Pennsylvanian age, those shown in figs. 4 and 5, extend from below *O* bed to the Cherokee shale. The pipe-slumps of post-Pennsylvanian age extend into the Cherokee shale, and a few that can be observed underground formed above *N* bed where the solution of limestone, largely in *M* bed, allowed the overlying horizons to slump.

In some pipe-slumps the zone of vertical-circular shearing and the pipe both contained ore which was

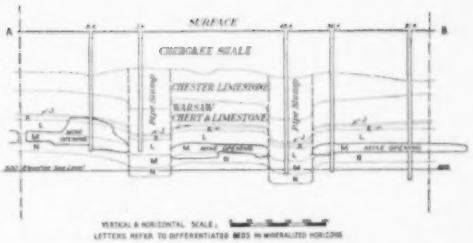


Fig. 4—Vertical section *A-B*, Webber mine, showing pipe slumps.

mined as one cylindrical orebody, whereas in some others the zone of vertical-circular shearing contained the ore which was mined as a circular ore body around the pipe which contained little or no ore.

Vertical-circular shearing occurs on small domes in the Picher field and probably is the result of regional deformation which forced the beds upward to form the domes. This contrasts with vertical-circular shearing resulting from slump of the beds into an opening. The forces in both cases acted in much the same manner; the beds were forced up into a dome in one case and forced down into a basin in the other. In both cases tension forces were set up around the areas of the pipes and the resolution of vertical compression forces with horizontal tension forces caused vertical-circular shearing.

Man-made pipe-slumps, one of which occurs at Negaunee, Mich.,[3] are similar to those that formed in nature. They are circular, extend several hundred feet to the surface, and formed over large undercut areas that were allowed to cave.

Chimney Ore Bodies

Several chimney ore bodies have been mined in the Picher field. As a rule they formed in pipes or on circular shearing and have been mined as continuous vertical orebodies 30 ft to 100 ft or more in height and are associated with structural deformation of different ages. While most of them are found extending above the main ore horizon of an area, a few extend below.

Residual Material and Breccia

Residual material such as chert nodules, limestone boulders, shale and clay, together with introduced material such as transported shale and a little sand collected over the bottoms of the large solution-made openings in the limestone beds. Where the overlying beds settled or collapsed into these openings, the residual and introduced material was compressed by the weight that crushed the loose chert and limestone boulders and squeezed the shale and clay into available openings. This crushed residual material of the premineral slumps became mineralized and the shale, clay and limestone were replaced by ore and gangue minerals which cemented the unreplaced shattered chert to form breccia.

Breccia, resulting from both regional and slump deformation, consists of chert fragments in a matrix of one of the gangue or ore minerals or combinations of them. Jasperoid is the most common of the matrix minerals and in the jasperoid area, fig. 6, is

WEBBER MINE near Treece Kas, in NE¼ SE¼ of Sec. 12-T35S R23E.
Contours on top of "N" Bed. Mine Workings in "M" Bed.

Fig. 5—Structure contour map for the top of *N* bed, Webber mine, showing locations of pipe slumps.

the exclusive matrix mineral except for calcite and occasional minor amounts of the ore minerals.

Calcite breccia is common because calcite filled openings in shattered zones and cemented broken rock fragments that resulted from deformation throughout the field at the close of the period of sulphide mineralization.

Under some of the slump areas there is an overlapping laterally of premineral and postmineral slumping. The residual material that was under slump blocks before and during the period of sulphide mineralization was mostly mineralized and changed to breccia. The residual material that collected in openings in the beds after the period of mineralization, is of course, not mineralized and remains unaltered.

Mineralization Maps

Mineralization maps, fig. 6, are made on plan maps of the underground workings and show the shearing and respective areas of mineralization in and around the shear zones. They are usually made on one mineralized limestone bed, but occasionally, when two or more contiguous limestone beds are mineralized, they are grouped and one map serves them all. Mineralization maps have been used in the office of G. M. Fowler at Joplin for many years, and maps published by the U. S. Geological Survey[8] under the direction of E. T. McKnight show dolomite areas in parts of the Picher field.

The mineralized zones are divided into three parts depending on the prominence of certain minerals in each part. These parts are referred to as the dolomite, the sphalerite-galena and the jasperoid areas on the mineralization maps.

The dolomite area occupies the inner part of the area of mineralization. It contains gray, coarsely crystalline dolomite that replaced the original limestone and also contains pink dolomite crystals that deposited in openings. Where the dolomite area is large or where mineralization was weak, the center of the area frequently contains much unreplaced limestone.

The sphalerite-galena area is the ore area and eventually is all mined. It lies between the dolomite and jasperoid areas but also contains both dolomite and jasperoid. It occupies most of the area within the mine workings in fig. 6.

The jasperoid area makes a fringe around the sphalerite-galena area, is the outer part of the mineralized area, and consists almost entirely of jasperoid and chert. It varies from a few feet to several hundred feet in width and its outer edge is bordered by limestone areas.

There are many places in the mines where the sphalerite and galena mineralization pinches out along a shear zone, but the dolomite continues on one side of the strongest shearing in the zone and the jasperoid continues on the other. By following the strong shearing that separates the gray dolomite and the jasperoid areas, the sphalerite-galena mineralization often comes in again in a short distance.

Minerals

Dolomite: Two types of dolomite, the gray and the pink, occur in the Picher mining field. The gray dolomite replaced limestone and residual material, whereas the pink dolomite was deposited in openings.

The gray dolomite is coarsely crystalline. Individual crystals are pale pink in color, and the interstices between the crystals contain dark gray residual material from the limestone that was replaced which gives the rock as a whole a gray appearance. Gray dolomite is the principal mineral in the dolomite areas, fig. 6, where it was deposited early in the period of mineralization. It is found in the sphalerite-galena areas, but does not occur in the jasperoid areas.

Pink dolomite was deposited as crystals in openings, following the deposition of much of the gray dolomite. It occurs mostly in the dolomite areas but was also deposited contemporaneously with sphalerite, galena, and jasperoid crystals in openings in the sphalerite-galena areas.

Jasperoid: Jasperoid is the most abundant mineral found in the Picher field that was deposited during the period of sulphide mineralization. It is found in all three areas. It was practically exclusive in replacing limestone and other material in the jasperoid area where the sphalerite-galena content is too low grade to mine. It was deposited contemporaneously with sphalerite and galena in the sphalerite-galena area, and in some intensely mineralized areas it replaced gray dolomite in the dolomite area late in the period of sulphide mineralization. The jasperoid that replaced other minerals varies in color from dark gray and dark

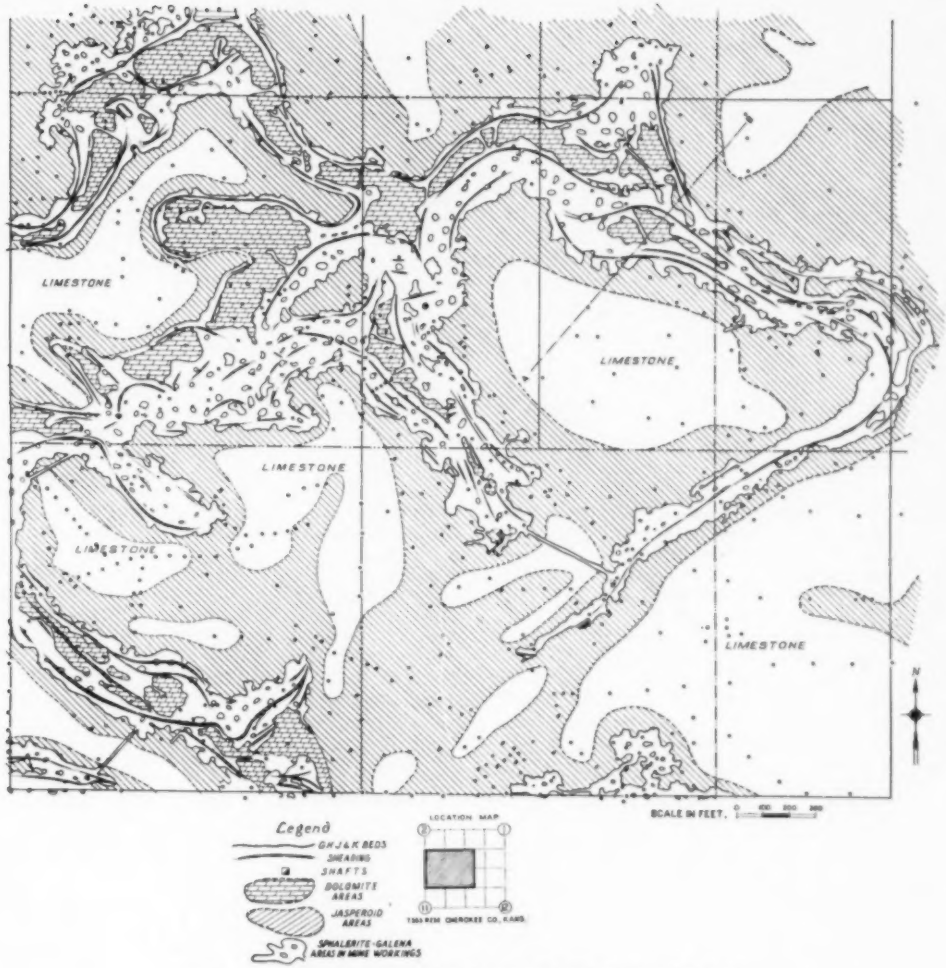


Fig. 6—Mineralization and shearing map for G-H, J and K beds.
Bilharz Mining Co. leases.
Eagle-Picher Mining and Smelting Co., Geological Department.

brown to black, consists of microcrystalline quartz crystals, and breaks with a splintery or conchoidal fracture. The jasperoid that deposited in openings consists of dark quartz-crystal-aggregates, in which the crystals, though minute, are visible with a hand lens, and in places are intergrown with sphalerite, galena and dolomite crystals.

A form of jasperoid often referred to as "sand" occurs in places where microcrystalline quartz crystals had only partly replaced the original limestone, and subsequent solution of the residual limestone by circulating underground waters left the quartz crystals as a porous, loosely knit mass in which fossil outlines are common. Much of this material resembles loose sandstone and crushes easily into sand-like particles, which is the reason it is referred to as sand. This type of jasperoid grades gradually to the more massive types of jasperoid where mineralization was more intense.

Sphalerite: Sphalerite is most plentiful along the shear zones that acted as channels for the ore-bearing mineralizing solutions. It diminishes in quantity away from the shear zones and disappears almost entirely toward the centers of many of the dolomite areas and toward the outer parts of the jasperoid areas. It replaced limestone, shale and dolomite but did not replace chert or jasperoid. It was contemporaneous with jasperoid and gray dolomite in replacing limestone, shale, and clay in the matrix of residual material, and later in the period of mineralization it was deposited as crystals in openings.

Galena: Galena, usually in small amounts, was deposited contemporaneously with the sphalerite. Much of the galena was deposited late in the period of sulphide mineralization. It occurs more abundantly in the outer edges of the sphalerite-galena areas and in the top bed of a mineralized zone, such

as *E* bed, fig. 7. It is found as crystal growths in openings on pink dolomite and sphalerite crystals. It replaced limestone, shale and dolomite but did not replace chert or jasperoid. It was contemporaneous with sphalerite, gray dolomite, and jasperoid in replacing limestone, shale, and clay in the matrix of residual material.

Marcasite: Marcasite is common but is not plentiful. It was deposited late in the period of sulphide mineralization. It is most commonly found in the outer edge of the sphalerite area and is often associated with shale. It occurs as crystals on other minerals in openings and often makes a coating that caps and completely covers other crystal aggregates. Massive marcasite is found occasionally in the outer edges of the ore bodies. Rarely, botryoidal marcasite is found coating calcite crystals.

Pyrite: Pyrite occurs sparingly and is usually found associated with marcasite. Both minerals were deposited at the close of the sulphide period of mineralization. A small amount of pyrite was deposited early in the period of sulphide mineralization and is more common in *J* bed.

Chalcopyrite: Chalcopyrite occurs in openings as small crystals commonly found on the inner edge of the sphalerite-galena area and in the dolomite area on sphalerite and dolomite crystals. It has been found sparingly in a few mines in the Picher field as massive chalcopyrite that was deposited contemporaneously with sphalerite.

Enargite:[13] Minute enargite crystals are found very sparingly on the inner edge of the sphalerite-galena areas and within the dolomite areas where it is found on pink dolomite and chalcopyrite crystals.

Quartz: Macroscopic quartz crystals were mostly deposited at the close of the period of sulphide mineralization. The microcrystalline quartz is the jasperoid and it was deposited throughout the period of mineralization. Clear quartz crystals ½ in. or more in thickness are found occasionally lining vugs and openings in *O* bed. Minute but visible quartz crystals commonly line small openings or fractures in chert beds and small quartz crystals are found as coatings on dolomite, jasperoid, sphalerite and galena.

Calcite: Calcite is the latest mineral that was deposited and is found as crystals lining caves and openings in the mineralized area. It is found as filling in shattered zones that formed in the ore bodies after the dolomite, jasperoid, and ore minerals had been deposited, and it is most abundant in the jasperoid area. Occasionally, marcasite crystals can be seen in calcite crystals.

Paragenesis of the Minerals

The relative order of deposition of the minerals in the Picher mining field as interpreted by the writer is shown in the paragenetic diagram, fig. 8.

There is a difference of opinion among geologists as to replaceability and age relations between jasperoid and the sulphide minerals sphalerite and galena. Many geologists are of the opinion that jasperoid, dolomite, sphalerite, and galena were deposited contemporaneously. Others[12, 14] who studied samples collected for the purpose of determining the paragenetic relations are of the opinion that the deposition of jasperoid began before any sulphides were deposited, that all the sulphides are younger

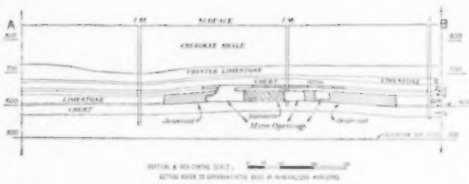


Fig. 7—Vertical section *A-B* showing mineralization in limestone horizons in Bilharz Mining Company's mines.

than the jasperoid, and that much of the sulphide material in the jasperoid occurs as automorphic crystals that replaced the jasperoid.

The writer believes that the dolomite, jasperoid, sphalerite, and galena were deposited contemporaneously and that the sphalerite did not replace the jasperoid for the following reasons:

First, Smith and Siebenthal[15] show from a microscopic study that the general process of replacement of the limestone by the jasperoid started with the deposition of a few scattered crystals of quartz in the limestone, and with increase in proportion of quartz the limestone decreased until the rock consisted of a granular aggregate of quartz crystals with scattered ragged grains of calcite, residual from the limestone, which finally disappeared. It was noted that sphalerite was found in the limestone with the first appearance of the quartz but was not found in limestone wholly free from quartz, which led to the conclusion that the sphalerite and quartz developed simultaneously. A micrograph of sphalerite in jasperoid and dolomite shows sphalerite crystals with ragged boundaries in the jasperoid, and crystal outlines in the dolomite, indicating interference of crystal growth by the jasperoid.

Second, jasperoid that was deposited in openings as quartz-crystal cluster-aggregates is often intergrown with small sphalerite crystals showing contemporaneous deposition, each capping the other as the growth proceeded.

Third, the glauconite particles which vary from pin point to sand grain size and which are scattered throughout the limestone in *J* bed were not replaced by the sulphide minerals, the gangue minerals, or the chert. Ore specimens that contain considerable glauconite show it concentrated around the sphalerite, galena and dolomite crystals, and scattered through the jasperoid, indicating that as the dolomite, sphalerite and galena crystals grew in the limestone, the glauconite was mostly pushed aside and the microscopic jasperoid crystals were subsequently deposited around it. A few particles of glauconite can occasionally be seen by magnification in the sphalerite and dolomite crystals but rarely in the galena crystals from *J* bed. If sphalerite had replaced jasperoid in *J* bed, it would contain considerable glauconite.

Fourth, as observed and agreed to by all geologists, the chert which was deposited first was not replaced by the ore or gangue minerals. Chert and jasperoid are chemically the same; both are silica. The difference is in crystallization, the chert being cryptocrystalline and the jasperoid microcrystalline. A chemical process that would affect one should have a similar effect on the other.

Fifth, shearing and shattering in the areas of mineralization continued throughout the period of

sphalerite mineralization, producing new fissures and openings which were repeatedly filled with ore minerals, sphalerite and galena, and gangue minerals, jasperoid and dolomite. This repetition of deposition of sphalerite and galena indicates that they were deposited both early and late during the period of mineralization.

From the foregoing described occurrences of the minerals, both as to time and place of their occurrence, it can be seen that they lend themselves to a classification that can be used to make mineralization maps, fig. 6. These maps separate the mineralized and unmineralized areas of the particular horizon, or horizons, being investigated and divide the mineralized area into areas characterized by the prominence and presence of certain minerals.

The objective of the mineralization maps is, of course, to help find ore bodies. When prospecting an area by churn-drilling, they outline the dolomite, sphalerite-galena and jasperoid areas and show whether or not the sphalerite-galena area has been thoroughly prospected; and in the mines where both the sphalerite and galena mineralization, on the trend of an ore body, becomes weak or plays out, they indicate the most favorable place in the mineralized area to prospect for them.

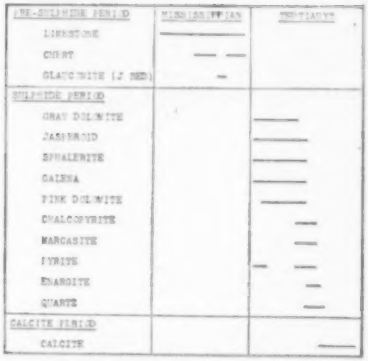


Fig. 8—Paragenetic diagram for minerals in the Picher Mining Field.

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George M. Fowler, with whom the writer was coauthor of several published papers on the Tri-State district, offered criticisms and suggestions, but may not necessarily agree with some of the statements.

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Mechanized Cutting and Face Stripping in the Ruhr

by R. R. Estill

THE rank of the Ruhr coal ranges from a high volatile bituminous coal to an anthracite, depending to some extent on the original depth of the seam. The average Ruhr coal corresponds to a soft bituminous American coal of a coking quality. The average thicknesses of individual coal seams being mined are also comparable (59 in. against 65 in. in the United States). However, consideration of seam conditions and mining conditions other than those just mentioned emphasizes differences rather than similarities with United States soft coal. In general, the Ruhr seams now being mined are much more folded and inclined than American seams. Dips of 20° and 30° are common in seams now being worked, and 30 pct of the coal reserves in the district are in seams dipping more than 35°. Only on the tops and bottoms of folds do we find rather flat coal seams. In addition to the folding there is extensive displacement by cross faulting plus a certain amount of strike faulting of an overthrust nature, which results locally in doubling or omission of seams. Because of the long history of mining in the Ruhr, nearly all coal lying near the surface has long since been mined out, and we find that the average depth of mining is at present about 2300 ft below the surface. Deep mining, folding, and faulting result in seam conditions requiring a great deal more roof support than one finds in American soft coal mines. In fact only in the anthracite district and the Rocky Mountain and Pacific coal fields do we find somewhat similar conditions. It is easy to say, therefore, that the problem of mechanization of coal cutting and loading in the German mines is quite different from that which we have so effectively met in America with our mobile cutters and loaders, duck bill loaders, and a room and pillar system of mining our drift and slope mines.

Partly because of more limited coal reserves, the traditional German mining system is largely the longwall method, which gives an almost complete coal recovery. Backfilling must be extensively practiced to protect the longwall faces, the over and underlying seams and workings, and especially the surface industrialized areas and barge canals. The German engineers have accordingly concentrated their efforts on the design of cutters, loaders, and conveyors suitable to longwall methods rather than room and pillar methods.

Undercutters with cutter bars like American models have been in use in the Ruhr since well before World War II. In 1941 they accounted for 8.5 pct of the production. This percentage, of course, includes coal which was undercut but nevertheless had to be broken down with air hammers or with explosives. The most common of these cutters is the

Fig. 1—Standard short wall bottom cutter (air operated).

Eickhoff Standard cutter (see fig. 1). This machine does about 95 pct of the undercutting in the Ruhr today, and is available with either compressed air or electrical power and in at least four different sizes. A variation of the cutter is this one with two cutter bars (fig. 2). At the end of 1947 about 200 of these machines and similar cutters were accounting for 13.2 pct of the total production, a production which was, however, only 60 pct of the 1941 production rate, so that the actual cutter tonnage was only up to a small amount over 1941.

In 1941 about 3 pct of the production was accounted for by shearing machines making their cut perpendicular to the longwall face. They were similar to those used in the States. These machines are today considered obsolete and now account for only 0.7 pct of the total production. They are located at only a few mines and at present do not seem to have much of a future in the Ruhr.

For the future, the Ruhr miner is looking forward to rather extensive mechanization of face work, with two major types of equipment being developed almost simultaneously. On one hand there is the development of cutter loaders for use in relatively hard coal. They represent the further extension of ideas developed after relatively long experience with the Eickhoff cutter. On the other hand there has been since 1942 an intense interest in the Ruhr in the development of face-stripping methods, particularly by the Kohlenhobel (coal plow) and its modification.

At the end of 1947 these cutter loaders, Kohlenhobels and scrapers together were actually accounting for only about 1.4 pct of total production while air hammers still broke 77.1 pct and as much as 1.2 pct was actually broken by hand picks. However,

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the Committee on Mechanization of the German Coal Mine Administration has recently estimated that it would ultimately be possible to reduce the tonnage of coal production by air hammers to less than 50 pct. Whether or not the air hammers will largely be replaced by stripping machines or by cutter loaders remains to be seen, but it is evident that the most promising machine is at the moment considered to be the Kohlenhobel. The Kohlenhobel (fig. 3) is quite successful in continuous soft coal seams with fairly good geologic conditions. The field of the cutter loaders is essentially in continuous harder coal seams. The air hammer and where necessary, drilling and blasting must continue to account for about half of the Ruhr production because so many seams are disturbed by faulting or other geologic phenomena. One member of the Committee of Mechanization has estimated eventually perhaps 25 pct of Ruhr coal could be mined by the Kohlenhobel and its modifications or by scrapers, and another 25 pct by cutter loaders, cutters and by blasting. However, in the case of coal seams of medium hardness, which constitute 20 to 30 pct of the total, there will be some competition between stripping and cutting machines.

Fig. 3—Standard Kohlenhobel (coal plow).

Cutter Loaders

For a closer look at the trends in design since the beginning of fully mechanized coal production in the Ruhr, we turn first to the cutter loaders since they were the first to be developed. In about 1938 and 1939 there was an industry-wide, concerted effort to achieve mechanization. The prize winning machine in a competition sponsored by the Bergbau-Verein (coal mining trade association) was the Eickhoff Rhine-Prussian cutter loader. This machine consists of a conventional cutter bar above which is a revolving toothed bar of readily adjustable height. In operation, the machine undercuts for about 18 in. and is then stopped to allow the toothed bar to be swung upward in an arc, thus breaking off a block of coal the height of the bed, the depth of the cut, and 18 in. thick. The paddle-type loader moves this coal into the conveyor. This model runs on top of the conveyor, and advances itself by means of a self-contained winch. Some of the other models of the Eickhoff have had two cutter bars, the upper one adjusted to the height needed to trim the roof. In some other models the lower cutter bar makes a sharp upward turn at its outward extremity (fig. 4). Such a model, combined with an upper cutter bar (fig. 5) can of course cut a block of coal completely free from the seam. As employed to date, however, including this latest arrangement, (fig. 6) certain difficulties have been experienced with these machines in that the lower cutter bar does not completely undercut the seam but instead leaves a wedge or rectangle of uncut coal on the footwall. The machine has therefore had to be supplemented by hand work. The newest models have a down-bent lower cutter bar.

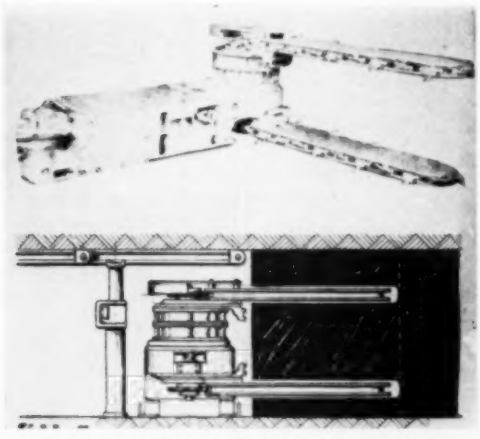

Fig. 2—Eichoff dual bar cutter (straight bars).

There are two ways of working a longwall face with one or more of the Eickhoff cutter loaders. The first method for working in soft coal uses two double chain flight conveyors. The forward or face conveyor is straddled by the machine, and conveys the coal away from the machine. The second conveyor is used only for backfill. The second method for working hard coal is to operate with only one conveyor, which unit must serve both for coal and backfill. In this method the machine does not straddle the conveyor.

These Eickhoff cutter loaders have required two machine operators, about five timber men, four backfill stowers, and four to five servicing the conveyors and winches. Efficiencies of 20 to 30 metric tons per face-man shift have been realized.

The Eickhoff cutter loaders have largely been designed to operate in seams thicker than 4½ ft. Another company, Deutsche Maschinenbau A. G., has been more active in designing cutter loaders for thinner seams, those down to about 3 ft in thickness. One of their DEMAG designs consists of the usual 6-ft cutter bar and paddle-type loader. The somewhat unusual factor is that it has a vertical shearer of adjustable height operating from the far end of the loader, like the English Meco-Moore machines which it closely resembles. This model skids itself between the face and the conveyor and does not leave a wedge of coal below the undercutter, but it cannot be used if the undercut coal does not immediately fall since there is no cutter at the hanging wall. Under the correct seam conditions these cutter loaders have shown comparably good performance, up to about 20 metric tons per manshift with an eight man crew.

In the field of cutter loaders the newest design is

Fig. 4—Cutter loader with turned up cutter bar.

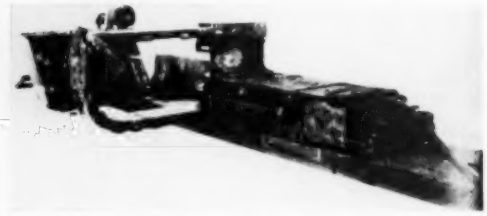

Fig. 5—Frame cutter loader.

the frame cutter (fig. 7). This frame cutter is still very much in the development stage, but experimental models are being watched with great interest by the Production Branch of the German Coal Mine Administration. The advantage of frame cutters and frame cutter loaders is considered to be that in hard coal, which will not fall by simple undercutting or even a double cut, or an undercut and a shear, the frame cutter is positive in that it completely surrounds its entire cut.

One of the Eickhoff machines under development designed to make a complete cut in a relatively thin seam of hard coal is a double-chain cutter with the roof cutter arm directly over the undercutter arm and with the far ends of both cutter arms connected by a revolving bar studded with spikes. Initial tests with this machine produced too much dust. However a modified version is to be tried in a different seam.

For cutters and cutter loaders in hard coal in steep seams, the Germans have turned to modifications of the foregoing types, since most of these have not been designed to work in seams steeper than 20°, or at the most 30°. One such cutter is currently being tried in the Dorstfeld mine. Here the dip of the seam is 60° to 70°. The cutter machine slides on its bottom and one side, its bottom being on the footwall and its side on the packing. Two ordinary chain-cutter arms do the cutting. The machine is moved along the face by winches on the cutter and in the upper gate road. In the present experiments this cutter works only from bottom to top, but it is planned to try it from top to bottom in order that packing operations can be carried on simultaneously. The angle of repose of the packing is of course a maximum to facilitate gravity movement of both coal and packing material.

Another cutter for a steep seam in hard coal is designed to operate under similar face conditions but instead of two chain-cutter arms it has one undercutting arm and one overlying spiked rotating bar with a cogwheel at the far end for breaking down the undercut coal.

Summarizing the German developments in cutters and cutter loaders for use in seams of harder coal, the only machine in widespread use in the Ruhr today is the Eickhoff chain undercutter. A large variety of other cutters and cutter loaders are under development or testing. The tonnage of coal suitable for these coming cutters and cutter loaders amounts ultimately to perhaps 25 pct of the total Ruhr output. It is therefore expected that in the next five or ten years several of the types of machines we have just seen may come into much more general use.

Face-Stripping Machines

There was little or no interest in the Ruhr area in the possibility of mechanized face stripping until 1942 when the Kohlenhobel (coal plow), fig. 3, at Ibbenburen was described and the results of experiments with it became generally known. Ibbenburen is a small hard coal district in Lower Saxony about 100 miles northeast of the Ruhr. The Kohlenhobel was given immediate attention after that in the Ruhr, and has today forged far ahead of the cutter loaders in total output. One reason for the interest is that it is suitable for the softer coal seams of a wide range in thickness. These seams are now largely worked by hand-held air hammers. Another reason for the popularity of the Kohlenhobel is its simplicity of construction and its lack of moving parts. Of course the most important reason for its popularity is that it enables the output per face-man shift to be approximately doubled. My last report indicated there are 30 to 35 Kohlenhobels in operation, and they produce about 4 pct of the total output.

Actually the development of the Kohlenhobel at Ibbenburen in 1941 and 1942 was a rediscovery of a machine and principle patented by Wissemann in the Ruhr in 1912. The Wissemann machine was not as simple in construction, nor was it reversible, otherwise its action was about the same. The Kohlenhobel is shown in detail in fig. 8 where the upper sketch is the side view from the conveyor side and the lower sketch is the plan view. As the plow is pulled along the longwall face, the coal is split from the solid by the face *d*. The continual movement of the plow lifts the bottom coal upward along this face, fracturing the seam above the plow and breaking it down while at the same time plowing the coal over onto the conveyor.

The use of the coal plow of course creates some unusual problems and procedures. The solution of one of these problems is shown in fig. 8 where the plow is revealed as actually two plows joined in the middle. Experimentation proved the necessity of permitting a minimum amount of vertical movement to allow for irregularities in the bottom. No hori-

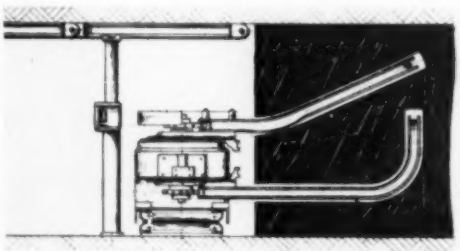

Fig. 6—Eichoff dual bar cutter (bent bars).

zontal movement is permissible, however. Other matters of design of the plow itself, particularly those of the shapes, sizes and angles of the cutting edges, are now standardized to conform with the nature of the coal, roof, floor, and thickness of the seam. Two sizes of plows are in general use. The smaller plow is 1 ft 4 in. high, about 20 ft long and 2 ft wide and is for coal seams up to 4 ft thick. The larger plow shown here is 2 ft high, about 20 ft long, 2 ft 4 in. wide and weighs about 3 metric tons. The height of the discharge is usually about 11 in., which is just sufficient to feed directly into the usual armored double-chain conveyor.

The depth of the cut in the softer coals where most of the plows are in use has been more or less standardized at about 1 ft, although it has been found desirable to make only a 6 or 8-in. cut in thick seams or in somewhat harder coal. The cutting edges are ordinarily made of a nonalloyed steel of 350 Bhn. It is only recently that experiments have been under way with Widia alloy of 600 to 700 Bhn. It has been found advisable to keep a number of spare cutter edges on hand at the face and to watch the cutter edge carefully, otherwise a broken edge may jam the conveyor or cut the cables. The blades are adjustable for height so that in case of a weak bottom the plowbottom can be made quite flat and in the case of a solid footwall the blades can be lowered to permit a small pressure on the bottom.

Another adjustment of the plow which is variable with bottom conditions is the height of the idler for the cable connection. The more solid the bottom, the higher can be the cable connection and the less tendency will there be for the plow to climb in the seam. If a plow is used on a flat seam, a climbing tendency is uncommon, but if a plow is pulled downhill in an inclined seam it will occasionally climb. For this reason a coal plow used in an inclined seam generally cuts only on the up trip and merely cleans its cut on the down trip. When a plow encounters a section of coal which is harder than the average in the seam, the practice is to run it back and forth until it has been able to chop its way through into softer coal.

There are several variations of the standard plow, most of them designed for thinner seams or for somewhat harder coal seams. One such plow newly designed is the Stufenhobel (step plow) which makes cuts in only one direction but at two elevations rather than one (fig. 9). The upper cut is made first, and its butting blade is preceded by four under-

Fig. 7—Experimental frame cutter.

cutting chisels, each extending the cut of the preceding chisel from 5 to 7 to 9 and finally 18-in. depths. The upper cut is at the level of the top of the conveyor. The lower knife scrapes the bottom and elevates the bottom coal by a ramp to conveyor level as in the conventional Kohlenhobel. The Stufenhobel is made by Deutsche Maschinenbau A. G. (DEMAG) and its first tests have been successful. Its present tests are being conducted in somewhat harder, thinner coal than that in which a conventional Kohlenhobel is usually employed.

The Flottmann Co. in 1945 brought out what they call the Schlaghobel (striking plow) which is very much like the conventional coal plow with the exception that four air hammers are built into each vertical cutting edge, to permit the plow to chop its way through harder coal seams (fig. 10). This first Flottmann Schlaghobel developed some design difficulties. The two lower hammers customarily become jammed, leaving only the two upper still hammering, as a result of which the Hobel showed a strong tendency to climb. The newer models are better in this respect but still have difficulties. Some German engineers consider that the depth of cut, 1 ft 8 in. is too great for hard coal.

A very similar activated plow constructed by the Hauscherr Co. has not had difficulties with its three air hammers jamming, but it has a high air consumption.

The earliest and perhaps the most successful of the plows with an activated cutting edge was the so-called Schnell-Hauer (quick miner) produced by the Westfalia Lunen Co. in late 1942. This model operated in only one direction, making a cut 2½ ft deep. A vibrating action was given to the cutter

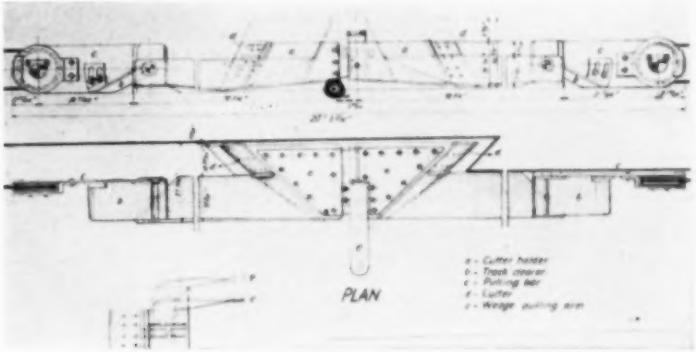

Fig. 8—Sectional view of the standard Kohlenhobel (coal plow).

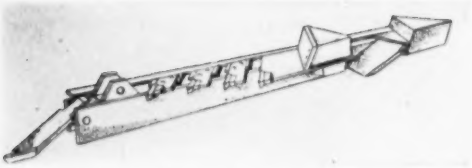

Fig. 9—Stufenhobel (step plow).

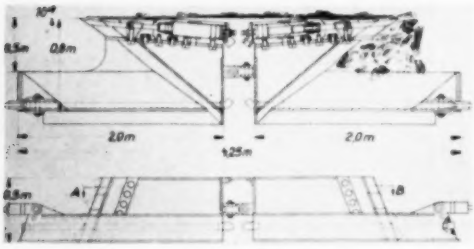

Fig. 10—Schlaghobel (striking plow).

edge, a series of chisels, by eccentrics, and the plow was able to reach a production of 360 tons per day in experiments in 1943. Encouraged by these experiments, the company developed a Schnellhobel designed to go in both directions (fig. 11). The eccentrics are located in the middle of the body (fig. 12). Unfortunately the results with this plow have been unsatisfactory so far and its further development is at the moment completely stopped. Its deep cut and the resulting large size of some of the broken coal caused both face-stripping and conveyor difficulties, and in addition the air consumption is too high.

Despite the unsatisfactory results today with coal plows of the activated-cutting-edge type, they are relatively certain in the future to receive further experimentation because of their promise in the harder coal seams.

Returning now to the conventional, proved, unactivated Kohlenhobel one can easily see that its use in a longwall face means that the mining practice with respect to conveyors, roof support, packing, and motor and winch installations is necessarily quite different from mining practice in a standard longwall face worked with the usual air hammers. An installation currently in use at the Friedrich Heinrich mine is shown in fig. 13. The coal plow is at the left end of the face. The winch positions are in the gateways, and the double-chain armored conveyor is illustrated in its position between the face or the Kohlenhobel on the one hand and the first series of props on the other. The proper anchoring of the winches, pullies, and deadmen has been a problem which has been fairly well worked out. With a cutting width of 1 ft and a rate of travel for the plow of 20 ft per min in coal of medium hardness, a winch must normally pull with a force of 5 to 8 tons and have a maximum pull of about 20 tons. The cables themselves must have a breaking strength of 50 to 60 tons. There has been some experimentation with chains rather than cables because of their lower elasticity and more positive pull.

The conveyors used along a coal plow face must not only have a large capacity but they must be capable of withstanding lateral pressures of 2 tons or more. This is because the force which holds the Kohlenhobel into the face is transmitted to the Kohlenhobel through the conveyor, or in another sense, the Kohlenhobel is confined to the face by the rigid emplacement of the conveyor. Perhaps the most common type of conveyor in the Ruhr today is the shaking conveyor, but for a Kohlenhobel face or, for that matter a cutter-loader face, the armored double-chain conveyor (fig. 14) with bottom return meets the requirement of low height and the necessary strength to be scraped against or ridden upon. The principal difficulty with this conveyor is its high power requirement. Experimentation with this conveyor is at the moment on rolling rather than sliding crossbars between the chains and also with a belt in place of the chains and flights.

In order to move the conveyors sidewise each time the Kohlenhobel has passed any given point and to anchor it there, several processes have been devised. The first and still most common system is that of air cylinders which press against the side of the conveyor every 20 to 25 ft. The next most popular system is the Ruckschlitten (sliding wedge). This sliding wedge is towed by a cable from the coal plow and presses on the conveyor as it slides along the front side of the front row of posts. Sometimes the sliding wedges are pulled by a special winch. This method has the advantage of maintaining a very straight longwall face. The disadvantage is that the wedge is inclined to knock out posts or push them back. Difficulties of this type have caused several mines to abandon this system.

Another system of advancing the conveyor was developed in one of the small sub-bituminous coal mines of the Bavarian Alps and has now been tried in the Ruhr with fair results. It consists of a simple pantograph braced against heavy cribbing. The miners can advance this simple lever system stepwise by hand tools.

Timbering and packing are considered essential to the safety of the Kohlenhobel front. Since there can be no post between the rear side of the conveyor and the coal face, a distance of about 3 ft 7 in., caps must be extended from the posts out over the conveyor and the plow space. The posts and caps usually required are steel or sometimes aluminum and are designed for easy attachment and detachment of the caps and for only the proper amount of give in the

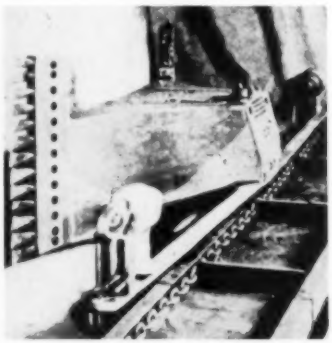

Fig. 11—Schnellhobel (quick miner).

posts. Backfilling at a Kohlenhobel face may be accomplished by dummy road packing if the roof will permit it. Solid packing is at present only feasible with pneumatic stowing since the double-chain conveyors do not permit slinger or hand stowing. Ruhr practice in pneumatic stowing is relatively efficient, and there is quite a bit of it done, approximately 12.9 pct.

The conventional Kohlenhobel and the other coal plows which we have discussed up to now are designed to be used in flat or nearly flat seams. The working of the coal plow in steep seams is dependent upon the development of other designs.

One such type of coal plow for steep seams has been designed by the Hausherr Co. This machine is lowered to its working position and anchored by compressed air jacks between the hanging and footwalls. A swinging arm with a plow head is set into operation by a compressed air cylinder. To date, as could be expected, its capacity is considered to be too small and its air consumption too large to be of real interest.

There were in the experimental stage several other types of plows for use in steep seams at the time the author left Germany in December 1948, but unfortunately he was unable to obtain any pertinent data prior to his departure.

Scrapers

Two interesting developments in face-stripping methods which are quite different from the previously-mentioned coal plows are the Ibbenburen scraper train and the scraper train development at the Minister Achenbach and the Julia mines. Fig. 15 is a schematic view of the Ibbenburen installation designed for the purpose of mining coal seams only 18 to 22 in. thick with a wet roof. A 50-m-long system of scraper boxes, at each end of which a Hobel is attached, is pulled backwards and forwards along the coal face by means of two winches. The individual boxes of the train have hinged flaps which glide over the coal when moving away from the middle gateway conveyor but which move the coal to the middle gateway when the direction of cutting is reversed. Each half of the longwall face was 164 ft long. No attempt was made to support the roof or to backfill. The advance of the train into the face was regulated by the position of the pulley blocks on the side gateways. The speed of the train was about 30 ft per min and the height of the cutter head and boxes was 16 in. The weight of the complete train was about 10 tons, and the pull on the rope while cutting was normally 3 to 4 tons with a maximum of about 10 tons. The face was developed in such a way that the pitch of the seam helped hold the scraper train against the coal face. For thicker seams the power requirement would be greatly increased and probably impracticable so that this scraper is especially for very thin seams. The particular seam at Ibbenburen was worked out by this method. The train is not now in use, but a future attempt would probably be made with longer faces since a large part of the objection to this method is the extraordinary gate road development necessary, especially as the gate roads have to be driven down the dip. The necessary rigidity of the train is another handicap.

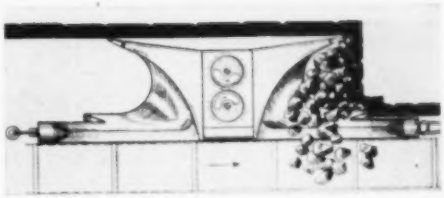

Fig. 12—Activated Kohlenhobel (coal plow).

Still another scraper train or scuttle scraper was in operation, the Minister Achenback. It has advantages over the Ibbenburen scraper train in that fewer boxes are required and it can work longer and thicker seams and is very flexible. With respect to the ordinary Kohlenhobel, the Minister Achenbach scraper train has the advantage in that it can be used where the seam is offset by small faults or is somewhat folded or otherwise irregular on its footwall. In this system the boxes are pulled up and down the coal face with a stroke of about 80 ft by means of a double-drum winch. As the scraper boxes move away from the gateway in which the conveyor is located, the flap ends of each scraper box glide over the loose coal, while on the trip toward the conveyor they move this coal to a position where the next scraper box will pick it up on the next inward trip. The individual scraper boxes at Minister Achenback are 8 ft long and 20 in. high with vertical cutting edges on each face side corner. The cutting depth is less than 3 in., but the travelling speed is comparatively high at over 3 ft per sec. The cable connections between boxes are arranged in such a way as to tend to pull the scrapers into the coal face when the cables are under tension. Guide rails are also used. Experience has shown that in the 5-ft seam at Minister Achenback which has been worked continuously by such a scraper train for several years, the coal undercut by the scraper train has generally come down by itself. The average production has been 250 tons per shift. A 120 hp winch is required for this coal face, but the installation costs

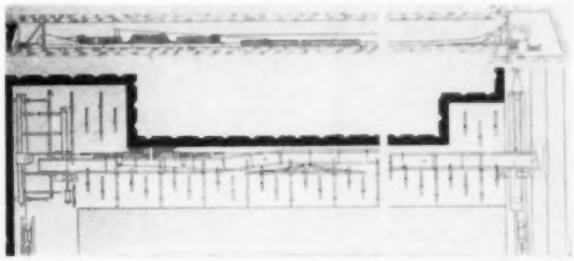

Fig. 13—Longwall face being worked with standard coal plow.

Fig. 14—View showing standard coal plow and armored conveyor.

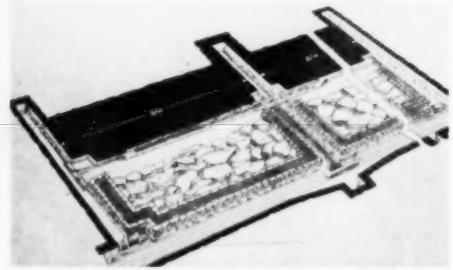

Fig. 15—Schematic view of scraper train loading coal.

are much lower than those for the Kohlenhobel with its expensive conveyor. And since this method is very adaptable to uneven seam conditions, to thin seams, etc., its use is being encouraged in other collieries such as Julia. There is no question, however, but that the standard Kohlenhobel, under the proper seam conditions, can give a much higher output.

Some efficiencies achieved in June and July 1948 by representative installations are given in table I.

Since the main subject of this paper has been German ideas and developments in coal cutters rather than machines for rock work, it will probably be sufficient to give the latter only a passing comment.

The Salzgitter loader, as many of you probably know, is copied almost directly from the Eimco shovel but is heavier. Most observations indicate that it is less efficient than the Eimco.

Another loader of a familiar type is that produced by Bergtechnik which resembles rather closely a track-mounted duck bill. The 3-m-wide duck bill is

Table I. Efficiencies of Face Stripping

(June to July 1948)

	Seam				
	Thickness	Dip	Face Length, Ft	Metric Tons Coal Per Operating Day	Man Shifts Per 100 Metric Tons[a]
Standard Kohlenhobel					
Best	4.3 ft	25 to 30°	790	365	14.8
Typical	5.3 ft	0 to 5°	610	444	23.3
Activated Edge Hobel	5.0 ft	3 to 6°	660	252	29.9
Stufenhobel	2.8 ft	18°	880	228	30.0
Scraper Train	1.7-2.1 ft	10 to 15°	750	167	32.5
Steep-seam Hobel	2.5 ft	72°	310	47	16.4

[a] For all labor, including packing, timbering, conveying.

the head of a short shaking-conveyor which feeds an inclined belt to the wagons. No side movement is possible. A loading capacity of one ton per minute is claimed. From the front of the duck bill to the discharge point of the loading belt the length is about 50 ft.

The use of drilling frames to advance tunnels in rock has not been popular in the Ruhr district. Some of the frames run on rails bolted to the steel arches of the drift, high enough so that the frame, when in a horizontal position, will clear the loading machine. The lower middle portions of some drill frames were designed for easy disassembling in order to pass the duck bill or other mechanical loader as required.

Another development being actively investigated is high-pressure rotary rock drilling to replace hammer drilling. The pressures used are a ton for a small diameter bit and up to three for larger bits. Good results in drilling speed in sandstones of 7 to 8 hardness were reported. For example, one such sandstone percussive boring including carbide-type bits gave a drilling rate of about 3 to 4 in. per min while the high-pressure rotary drill in the same sandstone gave 24 to 40 in. per min. Another advantage claimed for rotary drilling is reduced silicosis because of less dust and more water flushing. Another result is less physical strain on the miners.

These developments in rock work, like the developments in mechanized coal cutting, are not widely in use. Under the Nazi system, where most mining was done by cheap forced labor, it was not essential to divert steel and other necessary material to the saving of labor. Since the armistice, however, the Ruhr coal miner is a free citizen with the highest possible priorities for food, consumer goods, and housing and with as much dignity as it has been possible to extend to him. However, because of the general living and industrial conditions in Germany, it has not yet been possible to find for the mines all of the steel and other materials and supplies necessary to give the miner something with which to increase his efficiency, nor has it been possible to see to it that he receives and consumes an adequate amount of food. The coal miners' ration is the highest in Germany, and is comparable to that of heavy labor elsewhere, but the miner naturally shares his ration with his family.

Another difficulty is the present lack of adequate patent protection for new developments. When this deficiency is corrected, we may see some genuinely new postwar modifications of these cutters and of the necessary hoists, conveyors, steel props, etc. to make their operation possible.

It will take time and materials to achieve anything like the potential mechanization of coal cutting and loading; but since the Ruhr coal mining industry and the United States Military Government believe that increased efficiency underground will mean increased coal output which will in turn speed the general economic recovery of Western Europe, every encouragement is being given to increase the number of mechanized longwall faces in the Ruhr as rapidly as possible under the circumstances.

Acknowledgment

The author wishes to acknowledge the able assistance given by R. W. Wayland, formerly Special Advisor to the Chairman (United States) of the Anglo-American Coal Control of Germany, in compiling the data for this paper.

The Probability Theory of Wet Ball Milling and Its Application

by Elliott J. Roberts

The theory is developed that the tons ground through a given mesh per day in a wet ball mill is proportional to the percent plus that mesh in contact with the balls and the net power applied to the balls at this point. A grindability test is described.

DURING the course of a study of the fundamentals of classification in 1937, the need for a more basic understanding of the action of a ball mill became acute. Unless one knows how classification affects grinding, one cannot hope to effectively improve on classification.

The methods of evaluating grinding efficiency that depend on surface developed were studied but soon discarded for two reasons:

1. There was no apparent method which could be generally used to give a reliable figure for the actual new surface developed as a result of grinding. Subsequent papers have not changed this conclusion.
2. The practical evaluation of grinding in the main ore dressing applications was in terms of the percentage retained on a screen which passes 90 to 99 pct of the material and not in terms of surface area.

The Probability Theory

With the background of our experience in the field of closed-circuit grinding, together with the papers of Lennox,[1] Gow,[2] Gaudin,[3] Fahrenwald,[4] Coghill,[5] and others, the approach of the theoretical physicist was then tried. The thought was somewhat as follows:

When one grinds in a ball mill, a given expenditure of power leads either to a certain number of point to point blows per hp-hr or to a certain distance of line contact per hp-hr, depending on whether the action of the balls is considered to be cascading or rolling.

It is also assumed that the balls actually come together on each blow or during the roll. Then a volume of slurry will be covered per minute which is some function of the size of the particle being considered (see fig. 1). All particles coarser than this size will be reduced through this size. This volume of slurry contains a certain weight of ore, depending on the percent solids and the density of the solids.

If we fix the percent solids and the density of the solids and let w be this certain weight of ore in the volume covered, then, in mathematical terms, what we have just postulated is,

$$\frac{w}{dt} \propto \text{hp} \qquad \text{(a)}$$

If W is the total weight of ore present in the mill, then we can write,

$$\frac{W\ w}{W\ dt} \propto \text{hp} \qquad \text{(b)}$$

and if C is the cumulative percent plus the size chosen at the start of the time interval dt,

$$\frac{W\ w\ C}{dt\ W} \propto \text{hp} \times C \qquad \text{(c)}$$

But $\frac{wC}{100}$ is the weight plus the size chosen which at the close of time dt is finer than that size, and $\frac{wC}{W}$ is the decrease in the percent plus of the whole mass of ore or $-dC$. Then,

$$\frac{-W\ dC}{dt} \propto \text{hp} \times C. \qquad \text{(d)}$$

In other words, the mesh tons ground through a given size per unit of time is proportional to the hp and the percent plus the mesh. A crude analogy would be to picture a 1-ft-wide steam roller going down the road at 1 ft per sec. If we place one egg on the road per square foot, one egg will be smashed per second. If we place a dozen eggs per square foot, a dozen eggs will be crushed per second.

Similarly, if all the particles in w are plus the mesh, i.e., $C=100$, we should have a maximum rate of reduction. If only 10 pct of them are plus the mesh ($C=10$), we would have only one tenth the maximum rate; if only 1 pct are plus the mesh, the balls have a hard time finding anything to work on. This is where the term "probability theory" comes from. The chances of the balls crushing a particle through a given mesh depends directly on the concentration of particles coarser than this mesh in the general pulp in the mill.

Giving W the units of tons and dividing equation (d) through by W, we obtain

$$\frac{-dC}{dt} = k\ \frac{\text{hp}}{\text{ton}}\ C \qquad [1]$$

where k is a constant for any one size of particle, density of solid and moisture content of pulp.

Eq 1 is the rate equation for a first order reaction and says that the rate of decrease of the percent plus a given mesh with time is directly proportional to the hp per ton applied to the body of ore and to the percent plus the mesh in the ore mass as a whole. Since it is a differential equation, it only

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applies to an instantaneous condition, but it can be readily integrated to give eq 2.

$$\log_{10} \frac{Co}{Ct} = \frac{k}{2.3} \frac{\text{hp}}{\text{ton}} t \qquad [2]$$

where C_t = cumulative percent plus size at time t
C_o = cumulative percent plus size at zero time.

The Lake Shore staff[1] noted that "almost a constant percentage of the +28 micron material remaining in the feed is ground out in each 3-min interval" and presented data to support the observation but did not pursue the subject further. Their note is another way of saying what is expressed above and eq 1 is its mathematical form.

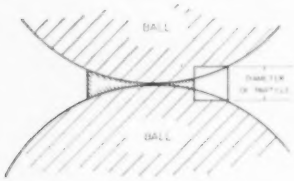

Fig. 1—Slurry volume covered by balls.

It is apparent from eq 2 that a semilogarithmic plot of C versus t should give a straight line if the equation holds. Fig. 2 shows a rather good agreement with the equation. Cumulative percent +150-mesh is plotted vertically on the log scale and time of grind is plotted horizontally. The mill was the laboratory mill described later and the charge was constant at 2 kg. Thus hp per ton is constant.

The data of fig. 2, line *a*, was obtained with a cement rock, ore *B*, by grinding charges of −20-mesh feed for 1, 2, 4, 8, and 16 min. A straight line through the first five points misses the 16-min point by 1.3 pct. A sample of the feed to *a* was classified at about 100-mesh removing 42.2 pct by weight of the sample. This brought the percent +150-mesh up to 86.5 pct. Grinds on this material as heads gave line *b*. The agreement of the individual points (circles) with the line is excellent.

If we let

$$\frac{k}{2.3} \times \frac{\text{hp}}{\text{ton}} = K \qquad [3]$$

then from [2]

$$K = \frac{\log_{10} \frac{C_o}{C_t}}{t} \qquad [4]$$

and it is apparent that

$$K = \frac{1}{t_a} \qquad [5]$$

where t_a = time at which $C_t = 0.1\ C_o$.

Applying this procedure, we find K for line *a* to be 0.055 and for line *b*, 0.052. This is in disagreement with eq 1, but by plotting the points of line *a* at t plus 3.8 min, we see that the discrepancy is not very great and the last point is practically on line *b*.

Practical Implications

To show what this relationship means in practical terms, let us compare the grams ground through 150-mesh in the first 2 min in each case. With line *a*, there were 240 g of −150-mesh produced in 2 min, while with line *b* there were 370 g or a ratio of 1.54 to 1 in favor of the classified product. A more dramatic illustration is to compare the time required to reduce 2 kg of feed to 5 pct +150-mesh. Curve *a* says 18.9 min. At 5 pct +150-mesh in the overflow 2 kg of feed *a* would give 990 g of +150-mesh to be reduced. At the 370 g per 2 min rate above determined by the upper part of line *b*, this would take 5.35 min or a ratio of 3.53 in power requirements for *a* vs *b*.

An actual closed-circuit test showed a falling off of K from 0.052 to 0.0472 before coming into balance, which would increase the time required to 5.9 min and cut the horsepower ratio to 3.2.

This is an unusually high ratio. It is generally more of the order of 2.0, and this lower ratio is due to a failure to obey eq 1. This is shown in fig. 3 wherein the results of two series of grinds on a gold ore (ore *A*) are plotted. Only the 200, 100, 48, and 28-mesh points are shown to avoid confusion. It is seen that the 200-mesh points fall on a straight line satisfactorily, but 100-mesh points of the 16 and 17.12 min grinds drop well below the line. This drop occurs before 8 min in the case of the 48-mesh points, so that a grind to 1.5 pct +48 would take only 14.3 min instead of 24.3 min by the straight line. To grind closed circuit from 76.0 pct +48 to 1.5 pct + at 3 to 1 circulating load, we must reduce the percent +48 by 74.5 ÷ 4 = 18.6 pct in each pass through the mill. In practice this means going in at 62.6 pct + and coming out at 44.0 pct +. The corresponding times on our straight line are 1.15 and 3.35 min or a net of 2.2 min per pass. Multiplying by 4 for 4 passes we have 8.8 min to grind 2 kg of feed to 1.5 pct +48-mesh vs 14.3 min open circuits, or a power ratio of 1.63 in favor of closed circuit.

In an actual laboratory closed-circuit test, K dropped from the value 0.07 shown in fig. 2 to 0.0625, which would cut the power ratio to 1.46.

In both cases mentioned, the K's at the mesh of separation had dropped about 10 pct when the closed circuit came into balance. This is by no means general. The K rose in another closed-circuit test on the gold ore at 1 to 1 circulating load ratio. This behavior lessened the beneficial effect of the higher circulating load. Other ores ground at 4 tons of mill discharge per ton of product ended up with a higher K than at the start and others were substantially constant. In all cases the comparisons were made at the same cumulative percent plus in the ball mill discharge, and eq 4 was used to evaluate the K's. Fig. 4 shows the results on the secondary grind of a very hard gold ore, (ore C), grinding −20-mesh feed with 2-in. balls. It is noted that 100 and 150-mesh results obey eq 2 very well, 65-mesh points curve downward as noted in fig. 3. The

**Table I. Grinding Tests on Ore *A*
Effect of Moisture**

Grind Time	0	2	2	2	2
Solids, Pct		75	70	60	50
Mesh	Cum. Pct +	Cum. Pct +	Cum. Pct +	Cum. Pct +	Cum. Pct +
6	8.8	0.6	1.6	1.5	1.4
8	26.4	2.8	5.4	5.9	6.2
10	39.3	6.9	10.1	11.0	11.0
14	50.1	13.6	16.8	18.3	19.5
20	59.0	23.1	25.8	28.0	30.4
28	65.8	33.0	35.3	38.2	41.1
35	71.4	43.1	45.2	47.8	51.0
48	76.0	52.2	53.4	56.5	59.4
65	79.8	59.7	61.5	63.8	66.4
100	82.8	67.2	67.8	69.9	72.0
150	85.5	71.8	73.1	75.0	76.8
200	87.4	76.3	77.4	79.0	80.7

200-mesh points lie on a straight line for the first 16 min, and then the slope of the line decreases slightly. The 325-mesh and 17.8 micron results give exaggerated forms of the 200-mesh line, the constants decreasing with time. On the same plot, denoted by circles, are the results of the fourth and fifth cycles of a closed-circuit grinding series at 4 to 1 circulating load ratio, making a 200-mesh separation. On the average, the points bracket the simple batch grind curves, showing no significant change in grinding behavior.

Effect of Moisture

According to the original hypothesis, the moisture content of the pulp in the mill should affect the grinding rate. To be consistent with the semilogarithmic behavior just demonstrated, the rate should be proportional to the volume percent solids in the mill. Of course, above a certain figure ball action will cease and grinding will stop.

To investigate this effect, four grinds were made on ore *A* at 75, 70, 60, and 50 weight percent solids equalling 0.5265, 0.4635, 0.3570, and 0.270 volume fraction solids respectively. Table I presents the screen analyses of these grinds while table II presents the results in the form of ratios of equivalent times. An enlarged plot of fig. 3 was made and the times required at 70 pct solids to equal the results at the various percent solids read off. Dividing by the 70 pct solids equivalent time, table II resulted. The ratios of the various volume percent solids is noted as well as its 0.65 power which gives a reasonable agreement with the time ratios.

Thus eq 1 and 2 may be rewritten:

$$\frac{-dC}{dt} = k' \frac{\text{hp}}{\text{ton}} V^{0.65} C \qquad [6]$$

$$\log_{10} \frac{C_o}{C_t} = \frac{k'}{2.3} \frac{\text{hp}}{\text{ton}} V^{0.65} t \qquad [7]$$

where V = volume fraction of solids in the mill

$$k' = \frac{k}{(0.468)^{0.65}} = 1.64k \qquad [8]$$

A volume fraction solids of 0.468 was chosen for reference because it corresponds to 70 wt pct solids of 2.65 sp gr which was found safe for the first few grinds (1, 2, 4 min) and corresponds closely to mill practice.

The author is not inferring that the volume factor in the above equations will hold for all ores or all meshes any more than the log relationship itself holds rigorously. He merely desires to illustrate the magnitude of the effect in one case and show

Table II. Effect of Pct Solids in Mill, Equivalent Time at 70 Pct Solids to Get Results at Other Pct Solids

Wt. Pct Solids	75	70	60	50
Mesh				
35	1.09 min	1.00 min	0.88 min	0.76 min
48	1.11	1.00	0.86	0.70
65	1.14	1.00	0.89	0.73
100	1.14	1.00	0.84	0.70
150	1.14	1.00	0.83	0.70
200	1.12	1.00	0.83	0.65
Avg	1.12	1.00	0.85	0.71
Ratio of volume fraction solids	1.135	1.00	0.772	0.583
(Vol fraction solids ratio)$^{0.65}$	1.086	1.00	0.845	0.704

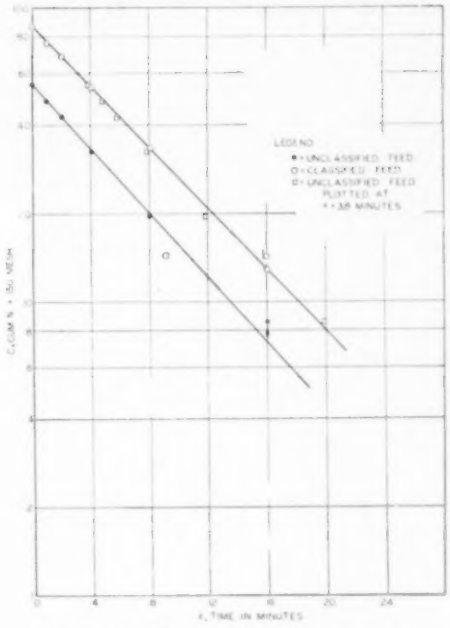

Fig. 2—Grinding tests, ore B, —20-mesh.

that the simple picture drawn at the start is incomplete and that the situation is really much more complicated.

In fact, if our original assumptions held rigorously, not only would the exponent of V in eq 7 be unity but k' would be the same for all ores and all ores would have the same intrinsic grindability (after allowance for density).

We know that this is not so and all of our deviations are probably due to the balls, not coming together each blow or during the roll and to the fact that different materials do have different shattering characteristics.

In other words, it is perhaps fortuitous that the agreement with eq 2 is as good as it is and therefore, all the equations are, in a sense, empirical, just as with 6 and 7.

Horsepower per Ton Factor

In view of Coghill's weight of evidence that the rate of grinding is proportional to the horsepower applied per ton, it seemed hardly necessary to go into this variable from the laboratory standpoint. However, sometimes the samples available for study were small, and it would help a great deal if the same sample could be used for the various grinds. Accordingly, a 2 kg sample was ground 2 min and 200 g removed for screen analyses; the 1800 g remaining were ground 1.8 min more to give the equivalent of a 4-min grind. This procedure was continued for 8, 16, and 32 min. Thus the last part of the 32-min grind was done on only 1200 g. To check this 32-min grind, another 2-kg sample was ground directly for 32 min. Table III shows that the results are essentially identical.

However, other tests have shown that on the coarse meshes discrepancies can occur. Table IV

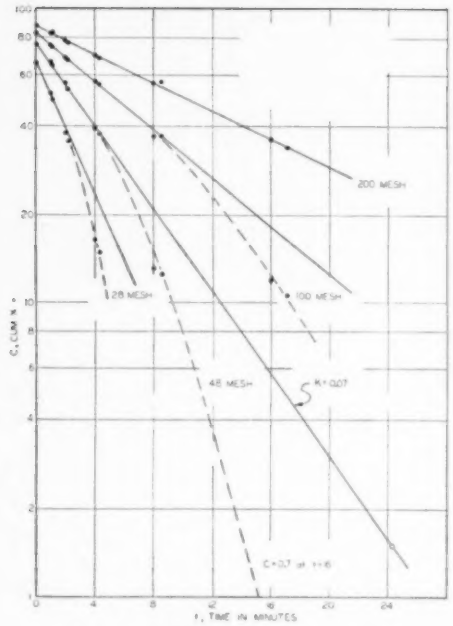

Fig. 3—Grinding tests, ore A, —4-mesh.

presents two pairs of grinds, which show that although at the fine meshes the results are equivalent whether 2 kg is ground for x min or 4 kg is ground for 2x min, on the coarse meshes the larger sample ground the longer time is slightly finer.

The Laboratory Mill

The other aspect of the horsepower per ton factor was whether the results in a small mill could be translated directly into behavior in mills of 6, 8, and 10-ft diam. The solution of this problem was fraught with innumerable difficulties. Accurate plant data had to be secured together with a representative sample of the feed. One of the biggest stumbling blocks encountered was the determination of the horsepower being drawn by the plant mill. Also for a direct comparison, the ball size in the two mills should be the same. This was made difficult to satisfy exactly because to keep the physical labor involved in making a test within bounds, it was decided to construct the mill from a section of 16-in. pipe measuring 15.4 in. ID by 6 in. wide.

It was felt that the ball action would be better with three rows of 2-in. balls than with larger balls. Furthermore, a charge of uniform balls would be much simpler to handle than a graded load. In addition, if the 2-in. uniform balls gave results equivalent to a 2½-in. graded load, a large number of plant feeds could be handled directly. Therefore it was decided to use a charge of 24.8 kg of 2-in. diam balls for comparative grinds. When rotated at 54.5 rpm, the loaded mill drew 0.150 hp, net, as defined by Gow.[3] Following Gow,[2] that, on the average:

Input hp to motor = gross hp = 1.33 x net hp [9]

the above set up was rated at 0.200 hp gross.

Thus a 1-min grind on a 2-kg sample is equivalent to:

$$\frac{2000}{4.4} \times \frac{0.200}{60} = 1.515 \text{ hp-hr per ton} \quad [10]$$

It was found that this uniform 2-in. ball would handle any —4-mesh feed put to it; i.e., the 6, 8, and 10-mesh particles were ground at such a rate that they would not build up unduly in a closed circuit operation.

In this respect the 2-in. load appears to be equivalent to at least a 2.5-in. graded load of forged balls.

On soft ores, feeds as coarse as —$\frac{3}{8}$ in. appear to grind satisfactorily, but, for comparative purposes, a —4-mesh feed with 10 pct +6-mesh was accepted as standard.

As indicated earlier, the standard charge was 2 kg at 46.8 pct solids by volume for coarse grinds, the moisture being increased as the charge thickened due to grinding.

Table III. Screen Analyses Cumulative Pct Plus 32 min Equivalent Grinds

Mesh	By Stages Ending with 1200 g	2-kg Charge Direct
65	0.4	0.5
100	3.1	3.8
150	10.5	11.6
200	20.8	22.8
325	36.7	37.3

Table IV. Screen Analyses Cumulative Pct Plus Comparing Effect of Size of Sample

Mesh	—4-Mesh Ore C, 2 in. Balls		—3-Mesh Soft Copper Ore 1½-in. Balls	
	2 kg for 8 min	4 kg for 15.5 min	2 kg for 2 min	4 kg for 4 min
20	1.3	0.1	0.8	0.8
28	3.9	1.9	2.0	0.9
35	11.5	9.4	5.7	4.9
48	24.3	23.5	14.2	11.3
65	38.2	37.5	25.9	23.8
100	49.0	49.9	37.5	36.4
150	57.7	58.3	49.8	49.0
200	64.4	65.0	59.0	58.8

Closed-Circuit Tests

To isolate the effect of mill size on the horsepower per ton factor, as distinct from the other variables, closed-circuit grinds should be made exactly duplicating the grind, circulating load, etc., observed in practice.

This can be done, and, in some of the work to be reported, has been closely approximated. However, it is an exacting and time-consuming procedure to get into complete balance at the plant circulating load and unless the feed size used is exactly the same, it is impossible to duplicate all of the conditions required.

The tests reported were really designed for another purpose, which required that the tons of mill discharge per ton of new feed be in the neighborhood of 4.0. Not all of the plant tests were run at this ratio. Some of the feeds were as coarse as —1 in. while the small mill necessitated a —4-mesh feed. In the laboratory tests also, if the new feed was finer than the classifier rake product, it was put into the classifier, which was not always done in the plant. All of these variations required some adjustment, which was done by the aid of the principles previously established, but both the unadjusted

and adjusted comparisons are given so that the magnitude of the correction can be seen.

The closed-circuit tests were carried out substantially as follows: A 2-kg charge of —4-mesh ore was ground the number of minutes estimated to give a circulating load of 3 to 1. It was then classified at the proper moisture content in a 3-ft-deep tube at a draw-off rate estimated to give the proper mesh of separation. After a few trials, this could be done quite accurately. The oversize was returned to the mill with new feed equivalent to that withdrawn in the overflow and the process repeated, adjusting the grinding time to get 500-g overflow. This was continued until the grams per minute reduced through the mesh of separation was substantially constant.

It was not always possible to obtain test data which coincided with the taking of the head sample. In most cases, this was not serious because of the uniformity of the ores but could well have affected the ore *E* data.

Comparative Results

Table V presents the comparative data. Essential data on the plant circuit is first given, ending with the hp-hr per ton actually used. Then for comparison is the laboratory tons ball mill discharge per ton of feed and the hp-hr per ton indicated by the laboratory data. Since many of the plant feeds were coarser than the laboratory feed, the laboratory horsepower figure is not the directly-observed result but was calculated as follows:

$$\text{hp-hr per ton} = \frac{(NF - O)}{100} \times \frac{2000}{R} \times 1.515 \quad [11]$$

where NF = Pct plus mesh of separation in plant feed
O = Pct plus mesh of separation in plant overflow
R = g ground through mesh of separation per minute of grind in the laboratory closed-circuit test

It is seen that the first part of the expression determines the number of minutes which would be required to reduce 2 kg of plant feed to the plant mesh of separation at the 0.20-hp rate, and 1.515 is the hp-hr per ton per minute of grind on 2 kg previously noted (eq 10).

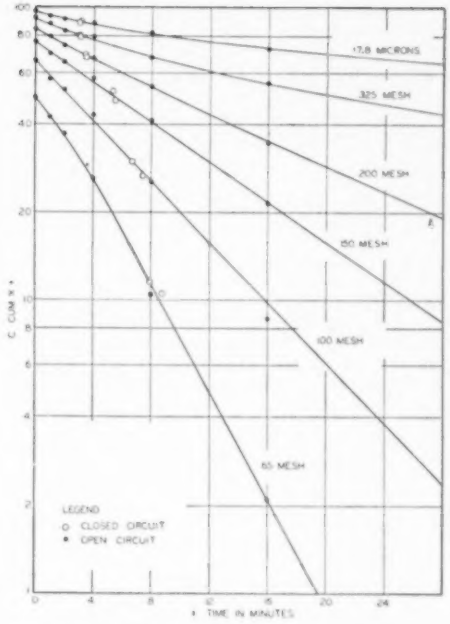

Fig. 4—Grinding tests, ore C, —20-mesh.

The hp-hr resulting from this calculation is thus the energy required *in the laboratory mill* to grind feed from the plant *NF* to the plant overflow fineness at the *laboratory circulating load*. Other qualifications were that the ball size was 2 in. uniform, that the feed was —4-mesh (or —20-mesh in some cases), and that the laboratory power figure contained a 25 pct allowance for motor, bearing, and windage losses.

Therefore, if plant feed, overflow, and circulating load conditions were matched perfectly and if the plant motor, bearing and windage losses were 25 pct; then, if the ratio between plant and laboratory hp-hr equals 1.00 for a plant make-up ball size of around 2½ in., the horsepower per ton factor in eqs 1 and 2 may be considered to hold independent of mill size.

Table V. Comparison of Laboratory Closed-Circuit Results with Plant Results

	Plant Data				Laboratory			
Plant	Grind Pct + Mesh	Make-up Ball Size in.	Tons BMD / Ton Feed	Hp-hr / Ton Feed	Tons BMD / Ton Feed	Hp-hr / Ton Feed	Hp-hr Ratio Plant / Lab.	Hp-hr Ratio Adjusted for C_{circ}
A	2.6 pct +48	2½	5.0	14.0	4.0	13.4	1.04	1.04
A	1.3 pct +48	2½	4.5	13.8	4.0	13.9	0.99	0.99
D	6.2 pct +65	2	1.8	10.2	4.5	9.2	1.11	0.90
E	1.4 pct +65	2	4.6	6.8	4.0	6.4	1.06	0.97
F	1.3 pct +35	3	4.5	4.4	5.3	3.3	1.33	0.97
F	3.7 pct +35	3	5.0	4.0	5.3	3.4	1.18	1.06
G	3.0 pct +48	4	3.7	12.6	3.3	11.0	1.15	1.20
C_1[a]	20 pct +35	3½	3.3	12.0	4.0	10.2	1.18	1.18
C_2[a]	1.4 pct +200	1¼	1.4	22.9	4.0	21.2[b]	1.08	0.60
B_1	1.6 pct +14	4	2.9	7.2	4.4	7.0	1.03	1.11
B_1	0.6 pct +14	4	2.0	8.3	4.4	7.1	1.17	1.11
B_2	13 pct +200	2	3.5	7.6	3.8	9.7[b]	0.78	0.75
B_2	10 pct +200	2	2.1	8.2	3.8	9.9[b]	0.83	0.77
A	3.4 pct +48	4	6.4	14.8	4.0	13.5	1.09	1.09
A	3.7 pct +48	4	5.5	14.6	4.0	13.5	1.08	1.08

[a] Subscript 1, primary; subscript 2, secondary.
[b] Grinds made on —20-mesh feed.

Table VI. Summary of Adjusted Hp Ratios

Ball Size in Plant	Plant	Adjusted Hp Ratio	Feed Size		Rel. Grindability	
			Plant	Laboratory	48-Mesh	200-Mesh
4 in.	G	1.20	— 1 in.	— 4-mesh	1.64	0.89
4 in.	B	1.11	— 1 in.	— 4-mesh	1.41	1.74
4 in.	A	1.09	— 4-mesh	— 4-mesh	1.40	1.10
3½ in.	C	1.18	— 2-mesh	— 4-mesh	1.00	1.00
3 in.	F	1.02	— 3-mesh	— 4-mesh	3.60	2.75
2½ in.	A	1.02	— 4-mesh	— 4-mesh	1.40	1.10
2 in.	E	0.97	— 8-mesh	— 4-mesh	2.38	1.86
2 in.	D	0.90	— 4-mesh	— 4-mesh	2.07	1.88
2 in.	B	0.76	—20-mesh	—20 -mesh	1.41	1.74
1¼ in.	C	0.60	—20-mesh	—20 -mesh	1.00	1.00

Table VII. Relative Grindability Standard K's and Percent Plus Ball Mill Discharge

Mesh	K	Percent Plus Ball Mill Discharge at Which K Was Used
14	0.0961	41.1
20	0.0818	41.7
28	0.0680	42.4
35	0.0575	43.4
48	0.0479	44.0
65	0.0398	45.7
100	0.0326	47.9
150	0.0264	51.0
200	0.0217	54.1
270	0.0174	57.5
325	0.0157	59.5
400	0.0142	61.9

Two tests on ore from plant A were the only ones where the ball size and other conditions were right, and here we find an average horsepower ratio of 1.015. The other two plant A tests with a 4-in. make-up ball gave an average ratio of 1.085, which is in line with the findings of other investigators on the effect of ball size.

In general, the other data of table V need some adjustment for circulating load. This was done by the use of the part of eq 1 which says that the rate of reduction at any moment is proportional to C. To use this relationship, we need an effective average value of C, C_{avg}, in the mill.

This C_{avg} is the logarithmic mean value of C,

$$C_{avg} = \frac{C_o - C_t}{2.3 \log_{10} (C_o/C_t)} \qquad [12]$$

zero time being considered when the feed enters the mill and time t when it leaves.

This logarithmic average is lower than the arithmetic average by an amount depending on the difference between C_o and C_t and approaches the arithmetic average only at high circulating loads.

The laboratory hp-hr per ton figures are corrected to the plant C_{avg} by taking

$$\frac{\text{Lab } C_{avg}}{\text{Plant } C_{avg}} \times \text{col. 7} \qquad [13]$$

Dividing this into col. 5 gives us col. 9 which is the same as:

$$\text{col. 9} = \text{col. 8} \frac{\text{Plant } C_{avg}}{\text{Lab } C_{avg}} \qquad [14]$$

Actually, in some cases a further adjustment was made for the change in K with percent plus in the ball mill discharge; i.e., for the deviation from the simple semilogarithmic relationship.

These adjusted ratios or conversion factors are placed in order of descending ball size in table VI together with data on the size of the plant and laboratory feeds.

It is obvious from the way the adjusted ratios follow the plant ball size that the hp per ton factor in eq 1 probably does hold and that these ratios are really ball-size factors. The data are neither sufficiently extensive nor accurate to permit of any generalizations on the relationship between ball size and factor at this time, however.

Grindability

If the K's for the different meshes (as determined directly by eq 4, or by plotting and eq 5) are compared, it is apparent that a "grindability" series will result. To be strictly accurate, the K's should be those resulting from closed-circuit tests but that would require a closed-circuit test at every mesh, which would be excessively time consuming. Therefore the K's were determined from simple batch grinds made on —4-mesh (10 pct +6-mesh) ore at 46.8 pct solids by volume. The K's for the different grinds were then plotted against percent plus the mesh in the ball mill discharge and the value of K at a definite percent plus in the ball mill discharge read off from the plot. These K's were divided by the corresponding K's determined on ore C and the result called the "relative grindability." Table VII gives these basic K values together with the standard ball mill discharge percentages at which they were read off. These ball mill discharge figures correspond closely to the ball mill discharge resulting from a 3 to 1 circulating load on —4-mesh feed on an average homogeneous ore.

Table VI carries the "relative grindabilities" of the ores involved at 48 and 200-mesh. We have found only one ore which gave a value of over 4 at 48-mesh and none which gave over 4 at 200-mesh.

It has been demonstrated that in a wet ball milling operation the rate of reduction through any mesh is roughly proportional to the percent plus that mesh.

It has also been demonstrated that the rate of reduction is substantially proportional to the hp per ton applied to the balls and independent of the size of the mill.

A factor has been presented for the effect of percent solids in the mill in one case and data showing the effect of ball and feed size are given.

A "relative grindability" procedure based on the above principles has been outlined.

Acknowledgments

The writer wishes to acknowledge gratefully the help and cooperation of his professional associates both at Westport and in the other offices; and to express his thanks to the various plant superintendents and their staffs for sending samples to Westport.

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Effects of Rod Mill Feed Size Reduction

by John J. Strohl and Henry J. Schwellenbach

This paper relates the results obtained by decreasing the size of feed to a rod mill screen circuit. As would be expected, added production was gained and a finer grind with no tonnage loss was made possible.

THE MacIntyre mill of the National Lead Co. has four parallel sections in which magnetic and gravity separations are made. Ore from a 5000-ton bin is fed by Hardinge constant weight feeders to a 6x12-ft open-end Marcy rod mill in each section. Rod mill discharge is bucket elevated to two 4x10-ft Tyrock screens whose oversize is returned to the rod mill. Screen undersize is pumped to Crockett separators for magnetite removal, and the Crockett tailings are sized and tabled to make an ilmenite concentrate. Minus 200-mesh material, overflowed by the Crockett and Dorrco sizers is thickened, deslimed, and concentrated by flotation.[1]

By 1948, the development of a successful slime flotation process and numerous improvements in the gravity circuit of the mill had increased the capacity of the available concentrating equipment. At the same time, improvements in the operating techniques of the dry crushing plant and improved feeding of the crusher by the mining department had appreciably increased the production rate per operating hour in that plant. Therefore, it was decided to take advantage of the added crusher capacity to produce a finer rod mill feed and in that way increase grinding capacity. At approximately the same time, finer screens were installed in the rod mill circuits to give finer grind and thus improve liberation of the magnetite and ilmenite from the gangue and from one another.

Prior to the first part of 1948, a ¾-in. limiting screen was used for sizing rod mill feed. The first reduction in crushing plant screen size was to ½ in. With this screen, the circulating load in the tertiary crushing circuit was too high and plant capacity dropped below a practical hourly rate. A 9/16-in.-sq opening screen, installed next, proved satisfac-

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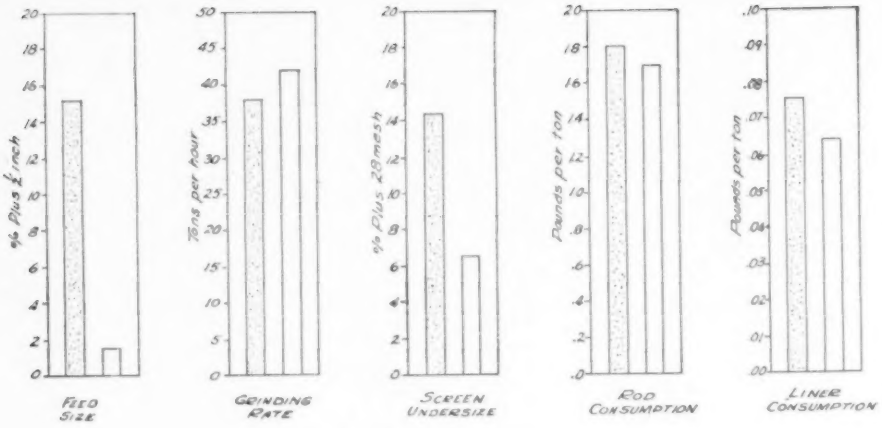

Fig. 1—Effect of rod mill size.

Dark bar is 3/4-in. limiting screen. Light bar is 9/16-in. limiting screen.

Table I. Comparative Screen Analysis of Rod Mill Feed

Screen Analysis of Products, Weight, Pct

Limiting Screen Dry Crushing	+1/2 in.	+3	+4	+14	—14
—3/4 in.	15.2	28.6	9.9	18.9	17.2
—9/16 in.	1.7	39.5	13.0	22.3	23.5

tory in dry weather but blinded badly with wet ore. Production rates are maintained by using a slotted screen in the winter and spring months.

After reducing mill feed size, the hourly grinding rate per mill averaged over a number of months before and following the size reduction, increased from 38.4 tons to 42.4 tons. A 230-hp input to each rod mill remained approximately constant over these months.

With a ¾-in. rod mill feed and 2½-in. rods, rod wear was extreme at the feed end of the mills and excessive shell liner wear was evident at the feed and discharge ends of the mills. In an effort to improve feed entry into the rod mass and decrease scouring wear of the rods and liners at the feed end of the mill, a shipment of 3½-in. rods was ordered. Their efficacy was not determined since the reduction of feed size was accomplished before these rods arrived. With a finer feed, the inordinate wear of the rods and shell liners at the feed ends of the mills decreased and liner life was greatly improved. Some 306,000 tons were ground by the only lining replaced since the feed change. The maximum tonnage ground by a set of liners prior to this time was 260,000. Rod consumption was lowered slightly.

Tables I and II give screen analyses of the two feeds. Unfortunately, more complete screen analyses of the two feeds are not available and it is impossible to evaluate the increase in surface of the finer feed as compared to ¾-in. rod mill feed. Some 11 pct[2] more surface in the screened rod mill product was produced after feed size and rod mill screen aperture were reduced.

Table II. Comparative Screen Analysis of Screen Undersize

Limiting Screen Dry Crushing	Rod Mill Screen	Screen Analysis of Products, Weight, Pct							
		+28	+35	+48	+65	+100	+150	+200	—200
3/4 in. feed	0.49 in.	14.04	13.81	15.33	13.01	11.55	8.67	5.76	17.83
9/16 in. feed	0.29 in.	6.45	12.23	15.61	14.69	12.51	11.14	7.67	19.70

The results of the two cases of rod mill feed size, as noted at MacIntyre, show that a finer rod mill feed will give a correspondingly greater grinding rate while grinding to a finer size, for better liberation, with a decrease in the rod and liner wear.

Fig. 1 shows graphically the variables in the two cases.

Acknowledgment

Appreciation is expressed to the entire staff and management of MacIntyre whose aid and participation made plant improvements possible, and to Frank R. Milliken, former Plant Manager, whose direction and advice has been invaluable to the advancement of the property.

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Laboratory Studies on Iron Ore Sintering and Testing

by F. M. Hamilton and H. F. Ameen

Results of a laboratory investigation of some sintering variables are presented. The effect on physical properties of mix component variation and rate of cooling of the sinter is illustrated by crush and tumbler test results and data of reducibility tests and microscopic examination of sinters are shown. The results of using solid additions in the mix to improve bed permeability are given.

SOME of the variables encountered in the sintering of iron ores and their effect on sinter properties have been investigated at the Jones and Laughlin Ore Research Laboratory in Michigan. Ores requiring agglomeration before blast furnace utilization may be classed as coarse, intermediate or fine. Coarse iron ore is considered similar to a −¼-in. Mesabi wash ore containing some 95 pct of its particles between ¼ in. and 100-mesh. Intermediate iron ore is considered similar to our New York State magnetic concentrates in which all the particles are −20-mesh and about 60 pct are +100-mesh. Fine ores are considered as being at least 100 pct −100-mesh. This paper will deal only with the two coarser classifications.

Most of the work has been done in a small experimental batch unit previously described in an AIME paper. Results being obtained form a basis for present and future trials in production sinter plants. It is expected that new production units that may be added in the future will benefit from laboratory studies. In general, the results reported here are obtained by laboratory study, but it is hoped they will be helpful in future plant practices and design of new plants.

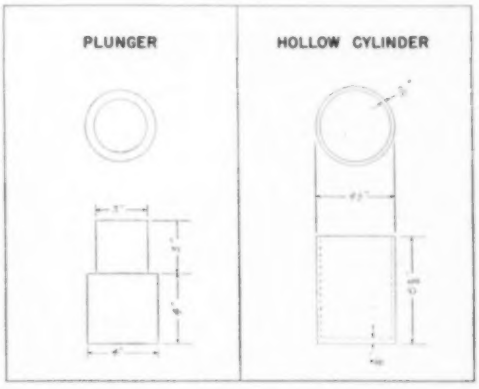

Fig. 1—Hollow cylinder and plunger, sinter crush test apparatus.

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Laboratory Equipment and Use

Experimental Sinter Machine: The laboratory sinter machine employed in the work consists of a chamber with 1 sq ft of grate area. It is mounted on

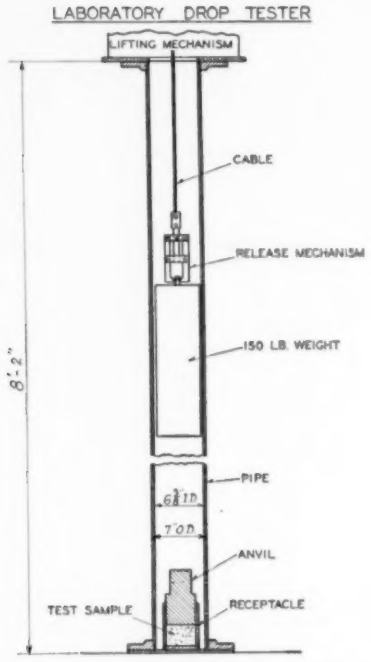

Fig. 2—Schematic diagram, sinter crush test apparatus.

trunnions and is rotatible to permit dumping. The chamber is connected through a 4-in. exhaust line to a suction fan. An orifice in the exhaust line and suitable taps to a U-tube permit gas flow measurements. A thermocouple in a well in the outlet of the sintering chamber is connected to a strip chart recorder and yields a continuous record of exhaust gas temperatures. The peak temperature is utilized as a reference point to measure the sintering time.

Physical Test Apparatus and Typical Results: Two methods are used for testing the physical properties of sinter. They are the result of an effort to find suitable methods of measuring these important properties. Many possible avenues were explored, including standard coke tests, impact crush tests, and pressure tests of various kinds. The methods found best for our work are crush and tumbler tests.

In the crush test, sinter to be tested is first sized by hand between 1 in. and ¼ in. It is then placed in a hollow cylinder (fig. 1) and a plunger or anvil inserted. Next, a weight of 150 lb is dropped on the plunger from a height of 53½ in. (fig. 2). Since the results obtained are only comparative between samples, the distance of drop is not too significant. For this reason, a mechanism already on hand was used for lifting and dropping the weight and 53½ in. was one of the increments readily obtained. After crushing, the material is removed and screen analysed. Distribution of the particle size is plotted (fig. 3) and serves as a comparison between different kinds of sinter.

Fig. 3 illustrates the results of crush tests on soft and hard laboratory sinters, along with those of a highly oxidized, friable magnetite sinter obtained from Sweden. Curve *A* represents the results on hard sinter, curve *B* the results on soft sinter, and curve *C* the results on the Swedish sinter. The figure shows appreciable differences between the curves of the different sinters. This test is a valuable tool in the study and examination of sinters.

The tumbler test is a modified version of the standard ASTM coke tumbler test. The apparatus used is identical to that of the coke test. However, a weight of 500 g of sinter sized between 1 in. and ¼ in. is used. The sized material for testing is placed in the tumbler apparatus and the drum rotated 500 revolutions at 24 rpm. Upon removal, the sinter is screen analysed and plotted as shown in fig. 4. Here again, curve *A* represents the results on hard sinter, curve *B* on the soft sinter, and curve *C* on friable Swedish sinter.

These testing machines have also been used to test pellets of New York State magnetite concentrates. One batch was designated hard, the other soft. Crush and tumbler tests were made to determine whether appreciable differences in results could be obtained. Fig. 5 shows considerable difference in crush properties between hard and soft pellets, and fig. 6 shows the same trend in tumbler results. Both show larger amounts of −60-mesh material (after testing) than the most friable sinters we have tested.

Fuel Content of Mix

The amount of carbonaceous fuel required to sinter an ore to a specified physical condition (as indicated by these tests) is less for Mesabi hematites than New York magnetites. The difference for hard sinters is approximately 1 pct. A sinter such as that of curve *A* in figs. 3 and 4 can be made from New York magnetites with about 6 pct carbon (exclusive of returns), whereas 5 pct carbon is sufficient when using a Mesabi hematite. For any particular mix, fuel content is a very important factor in determining the physical and mineralogical characteristics of the sinter produced. These charac-

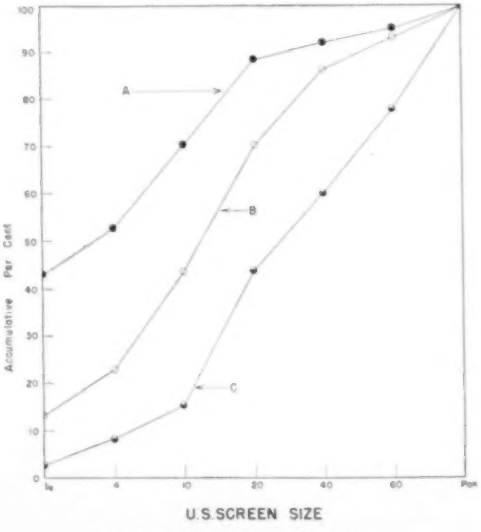

Fig. 3—Crush test results, sinter.

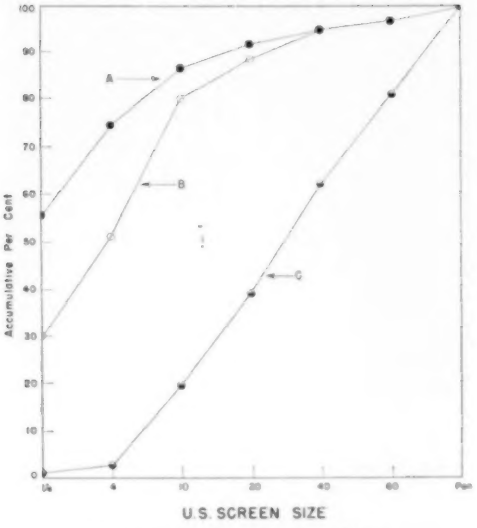

Fig. 4—Tumbler test results, sinter.

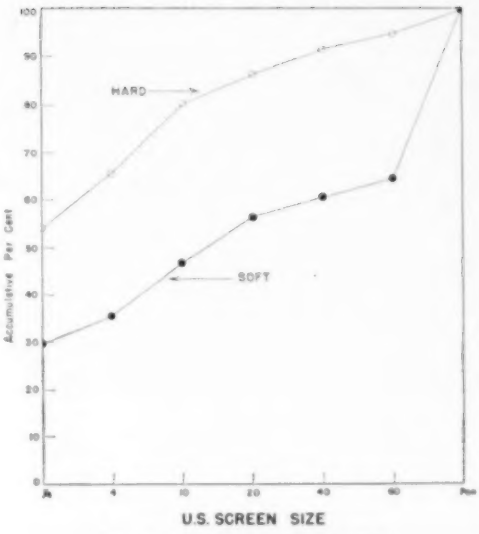

Fig. 5—Crush test results, pellets.

teristics have been and are now the subject of both physical and microscopic investigation.

Microscopic Examinations: In the microscopic work, the sinter samples are generally prepared as polished specimens and examined under reflected light. From this work it has been possible to determine the relationship of the different microconstituents, but as yet it has not been possible to determine the exact nature of the various silicates that are formed.

Examination of the different samples revealed that the material is composed of three major groups:

1. Unaltered fragments of the original ore minerals consisting of hematite, magnetite, and gangue.
2. Slag or binder material composed of artificial fayalite and amorphous glass.
3. Secondary iron oxides consisting of hematite and magnetite that have been dissolved and reprecipitated from the slag.

A polished section of hard magnetite sinter shows numerous medium sized equiaxed grains of magnetite surrounded by ribbons of silicate binder (fig. 7). The binder appears to be very similar to that observed in softer magnetite sinter. A few massive areas of silicate binder containing reprecipitated magnetite can also be observed.

Fig. 8 shows a soft magnetite sinter containing large grains of magnetite between which are dispersed small grains and ribbons of silicate binder. A few massive areas of silicate binders are present within which are stringers of dark-gray silicate. Numerous needles of hematite have been observed within the grains of magnetite.

A photomicrograph of a hard hematite sinter reveals small equiaxed grains of magnetite surrounded by slag (fig. 9). Note the very uniform distribution of magnetite within the slag.

A sample of soft hematite sinter shows it to be composed of fine grains of magnetite surrounded by thin ribbons of slag (fig. 10). Numerous massive areas of a eutectic-type structure may be seen.

Fig. 11 is a photomicrograph of a very soft and friable sinter made from Swedish magnetic concentrates. This material has hematite as the major microconstituent with some magnetite also present. The slag is noncrystalline and does not appear to coat the grains of iron oxide as in the other sinters. This is the sinter shown on curve *C* of figs. 3 and 4.

A number of facts are evident concerning the relationship of the microstructure of sinters and their physical properties. A significant factor is that con-

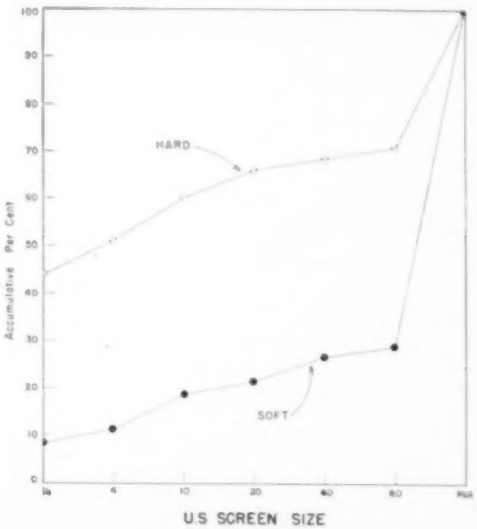

Fig. 6—Tumbler test results, pellets.

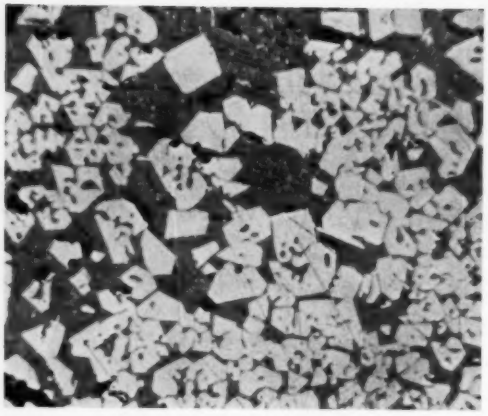

Fig. 7—Hard magnetite sinter. Light gray grains are magnetite, medium gray areas are silicate binders. (Air cooled.) X100.

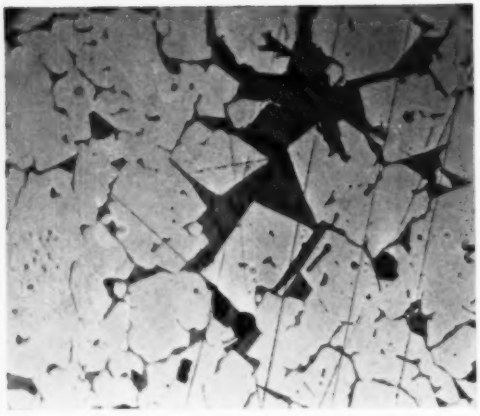

Fig. 8—Soft magnetite sinter. Light gray areas are magnetite, medium gray areas are silicate binder. Note dark stringer within grains of silicate binder. (Air cooled.) X100.

Fig. 9—Hard hematite sinter. White areas are magnetite, light gray areas are slag. (Air cooled.) X200.

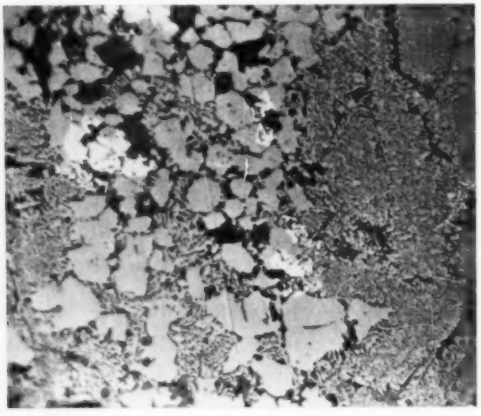

Fig. 10—Soft hematite sinter. White areas are hematite, light gray areas are magnetite, dark gray areas are slag. (Air cooled.) X325.

siderable solution and reprecipitation of the iron oxides takes place. It appears that upon burning the sinter mix, a considerable quantity of iron oxides are taken into solution by the molten silicates. Upon cooling to the melting point of magnetite, nuclei of magnetite begin to precipitate out as primary axes. As the process proceeds, with continued cooling, the spaces between the axes are filled, thus forming small grains of magnetite surrounded by molten slag. This process continues until the freezing point of the slag is reached whereupon the slag solidifies around the grains of magnetite. All of the original silicates do not become molten and all of the iron oxides are not taken into solution. However, the unfused particles are mechanically trapped within the sinter proper.

The predominant effect of carbon in the sinter mix (other than providing heat units) is chemical reduction. When the carbon content is lowered, reduction is lessened and/or more oxidation occurs.

It also appears that the greater the amount of carbon used and burned, the greater will be the amount of glass-like material. This glass-like material tends to enamel the surfaces of the iron minerals, acts as a binding agent, and probably renders it more difficult to reduce.

Reducibility Tests: Previous work by other investigators has shown that hard sinters are more difficult to reduce than soft sinters.[2, 3] Investigations with our hard and soft magnetite and hematite sinters show similar results (fig. 12). It was found that soft sinters show a higher reduction rate than their parent ores, whereas hard sinters show a lower rate. Also, New York magnetites exhibit slower reduction rates than Mesabi hematites.

Based on plant experience within our Corporation, we find that blast furnace operators are not too concerned with the better reducibility of softer sinters. Their main requirement is a sinter hard enough to withstand reasonably well the wear and

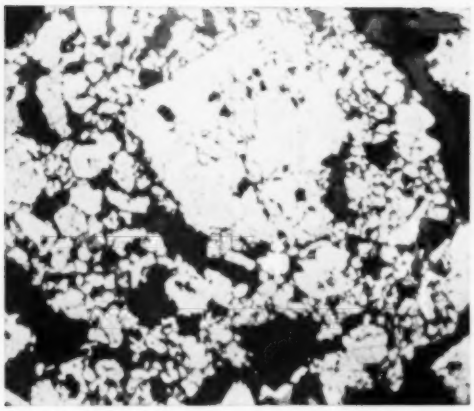

Fig. 11—Swedish sinter. White areas are hematite, light gray areas are magnetite, dark gray areas are slag. X60.

tear involved in transporting it from sinter plant to blast furnace skip.

Water Content

Sinter mixes have an optimum water content value for maximum flow of air through the bed. Small deviations from this optimum value result in appreciable decreases in airflow. Larger deviations result in a mix that will not sinter because it is too wet, or a mix that is partially pulled through the grate bars because it is too dry. The optimum value for any mix varies with the particle size distribution and the nature of the ore.

The effect of various water contents on airflow values in two sinter mixes, one composed of a coarse hematite, the other composed of a −20-mesh magnetic concentrate are shown in fig. 13. The data were obtained by making mixes of varying water contents, placing in the experimental sinter machine, and measuring the flow of air through the mixes at a definite vacuum. The mixes were not ignited to obtain these data. It can be seen that the hematite gives the best airflow at 10.5 pct moisture and the magnetite at 7.4 pct moisture. When the mixes were ignited and sintered, the fastest sintering rates were obtained from mixes containing amounts of moisture close to the optimum previously determined by airflow tests.

In the operation of tonnage sinter plants, close control of water in the mix is difficult. Plant application of moisture control is hindered by (1) variations in moisture content of raw materials, (2) large time lag between bins and swinging spout, (3) use of crude returns of varying size and temperature, and (4) lack of availability of a fast recording or indicating continuous moisture meter.

Solid Additions to the Mix

A bed of ore being sintered will have at least four general zones or layers of materials as follows: (1) Sinter layer, (2) fusion layer, (3) dried-out layer, (4) mix layer. Each layer will vary considerably in its permeability, thereby affecting the sintering rate. Coarse ores such as undersize wash ores exhibit a much higher sintering rate than intermediate size ores. In our plants there is a 25 pct difference in sintering rate between a coarse ore with about 95 pct of its particles between ¼-in. and 100-mesh and an intermediate size concentrate with some 40 pct −100-mesh. Attention, therefore, has been directed towards the possibility of improving the overall permeability of the sinter bed by improving the permeability of the zones or layers of material present during the sintering operation.

The sinter layer, as would be expected, is the most permeable portion of the bed. Airflow readings on our experimental machine normally increase 300 pct to 400 pct from ignition to heat completion indicating that little could be accomplished with this layer.

Hartman[1] reports that basic additions to mixes of lean German ores have lowered the temperature of the fusion zone and increased the sintering rate. Work on our ores has not been successful along this line.

Examination of partially sintered heats shows that in our ores the dried narrow layer between the fusion zone and the mix appears to retain its structure well and we doubt that it is a major factor in retarding the sintering rate.

The layer of mix between the dried layer and the grate bars appears to be the main obstacle to an acceptable sintering rate in processing our intermediate size ore, and some progress has been made in alleviating this condition.

In addition to the layers of mix discussed above, there is the obstruction to airflow caused by the grate bars. Of necessity, the ratio of the solid area of grate bars to open area must be large in order to provide sufficient structural strength and resistance to excess burning and warping and to prevent the sucking through of large amounts of fines.

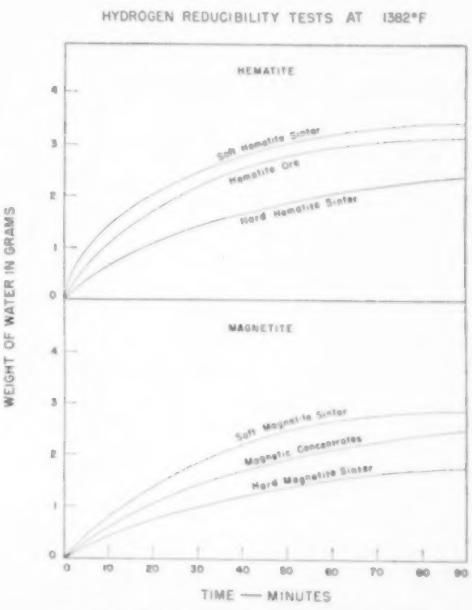

Fig. 12—Hydrogen reducibility tests, sinter.

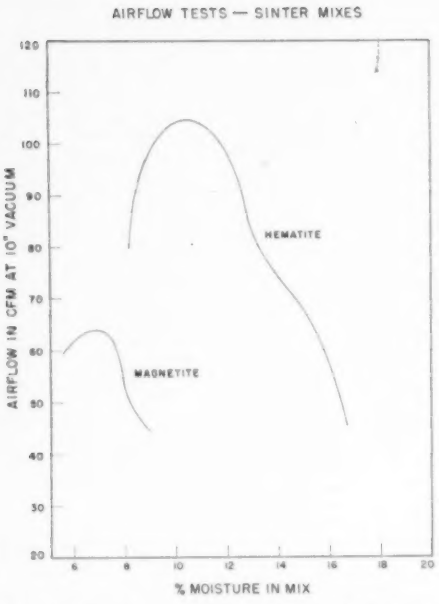

Fig. 13—Effect of water content on mix permeability.

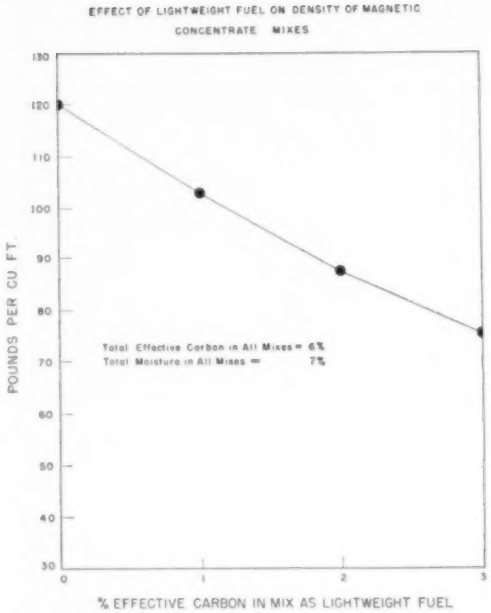
EFFECT OF LIGHTWEIGHT FUEL ON DENSITY OF MAGNETIC CONCENTRATE MIXES
POUNDS PER CU FT
Total Effective Carbon in All Mixes = 6%
Total Moisture in All Mixes = 7%
% EFFECTIVE CARBON IN MIX AS LIGHTWEIGHT FUEL

Fig. 14—Effect of sized sawdust on mix density.

In an effort to find out whether increased open area of the grate bars would aid materially in promoting airflow through the mix, a set of grate bars was designed and constructed for the experimental machine that increased the open area from a normal 23.5 pct to 42 pct. Subsequent tests on intermediate size ores showed no improvement in airflow during sintering.

Work on the mix layer itself has included studies on the effects of additions to the mix of (1) prepared agglomerates, (2) returns, and (3) lightweight fuel.

Prepared Agglomerates: In this part of the investigation, parts of normal mixes were removed and reformed as granules, briquettes or pellets. The agglomerates formed were then combined with the mix and sintered. In most cases the part removed was the −100-mesh fraction. In preparing the granules and briquettes, a number of binders were tried until one was found that enabled an agglomerate to withstand breakup until sintering could occur.

Fuel (as coke breeze) was added in various proportions to both the agglomerates and the mixes.

The results of these experiments were unfavorable as far as increasing the sintering rate was concerned. In some cases, particularly when attempting to sinter a mix with pellets, the mix showed improved permeability before ignition as indicated by airflow values higher than those usually obtained. After ignition, however, airflow values dropped markedly.

Examination of sinters made with prepared agglomerates reveals the difficulty of completely fusing briquettes or granules into sinter. Where fusion is not complete, the sulphur removal efficiency is poor. The magnetic concentrates tested contain 0.250 pct to 0.300 pct sulphur and it is necessary to remove most of the sulphur to make a suitable blast furnace feed. One of the experimental heats made with prepared agglomerates showed a sulphur content of 0.043 pct in a completely fused section and a sulphur content of 0.106 pct in a partly fused section.

Returns: Returns larger than about $\frac{1}{8}$ in. are of little or no value in promoting bed permeability and are a waste of good blast furnace feed. The problem seems to be determining the overall economics of operating a production sinter plant with completely processed returns.

Laboratory work shows that it is possible to prepare a mix with permeability equal to a normal mix by using less returns, if the returns are sized. Part or all of the returns larger than $\frac{1}{8}$ in. could be reduced to a suitable size. Any remaining material could be added to the sinter production of the plant or utilized as a bedding material. Economic gains might be obtained from increased production due to improved bed permeability and/or by a reduction in the size of the circulating load.

The use of sized returns has been tried and abandoned by at least one other company[5] on economic grounds. It is hoped that further efforts along these lines will be fruitful.

Lightweight Fuel: The addition of fuels with a lower specific gravity than that of coal or coke is quite helpful in promoting the bed permeability of intermediate size ores. Materials such as sized sawdust, sized agglomerates made from fine sawdust, and buckwheat hulls have been used successfully. Shavings and peat moss have been tried without success with our magnetic concentrates.

The upper and lower size limits of the particles of the best lightweight fuel are critical and at the same time are difficult to sharply define. Above

about No. 3 mesh the particles are too large to be completely burned in the relatively short time cycle of the process. Somewhere between 28-mesh and 48-mesh these light fuel particles start to become detrimental to the permeability of the mix. Too much fine material of this kind robs the mix of its normal moisture, clogs air cavities, and causes the production of a nonuniform product. However, when the particle size is controlled, it is possible to greatly accelerate the sintering rate of our magnetite concentrates. It is possible to increase the bed permeability of the intermediate size ore so that it will sinter at a faster rate than a coarse hematite ore.

The effect of sized sawdust additions to a magnetic concentrate mix in reducing the weight per cubic foot is shown in fig. 14. The total effective carbon in all mixes was 6 pct, part of the carbon being supplied by the sized sawdust and part by coke breeze.

With 0 pct carbon as lightweight fuel, the density of the mix is 120 lb per cu ft. At 1 pct, this drops to 103 lb, at 2 pct the value is 88 lb, and at 3 pct the weight is 76 lb. For comparison, a coarse hematite ore mix will have a density of approximately 95 lb per cu ft. The sintering rate does not necessarily increase proportionally as the density of the mix decreases, but the effect is quite pronounced.

Effect of Cooling Rate on Sinter

Water cooling of sinter, compared to air cooling, has been the subject of some discussion in the last few years. Saussaman[14] in 1948 reported that air cooling of sinter had reduced the amount of $-\frac{1}{8}$-in. sinter from 23 pct to between 3 pct and 8 pct at the Kaiser Plant in California. Tests in our plants and laboratory show that a decrease in the degree of drastic cooling results in improvements in this respect.

Figs. 15 and 16 show the effect of water and air quenching on a magnetic plant sinter, as revealed by the laboratory crush and tumbler tests. The sample was obtained by removing a large piece of sinter from one of the pellets before it was normally discharged. The sample was broken into two pieces. One piece was air cooled and the other quenched with a stream of water from a hose.

A pronounced difference in crush and tumbler results between the two differently processed sinters. These results represent extreme conditions of cooling not usually encountered in commercial sinters. However, the spread between curves is large in both cases and undoubtedly represents a major change in physical properties.

The differences between water-cooled and air-cooled sinters has been further studied with the microscope on experimental laboratory heats. Hard and soft sinters made from both coarse hematite and intermediate size magnetite were examined. An examination of the microstructures of two water-cooled magnetite sinters (figs. 17 and 18) revealed the following:

1. Watercooling alters the microstructure of magnetite sinter more drastically than it does that of hematite sinter.

2. More silicate binder is present in water-cooled sinter than in air-cooled sinter.

3. Silicate binder is more massive in water-cooled samples than in air-cooled samples.

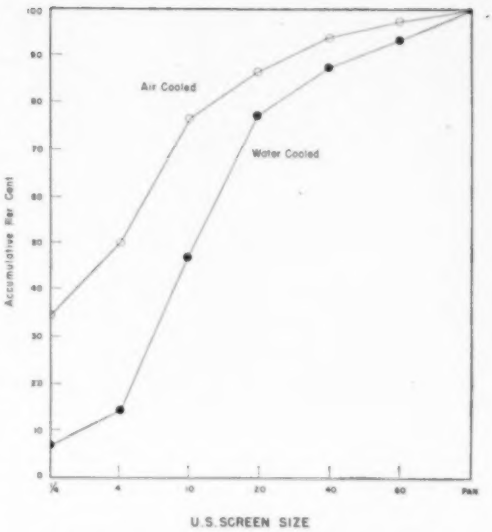

Fig. 15—Crush test results, water and air-cooled sinter.

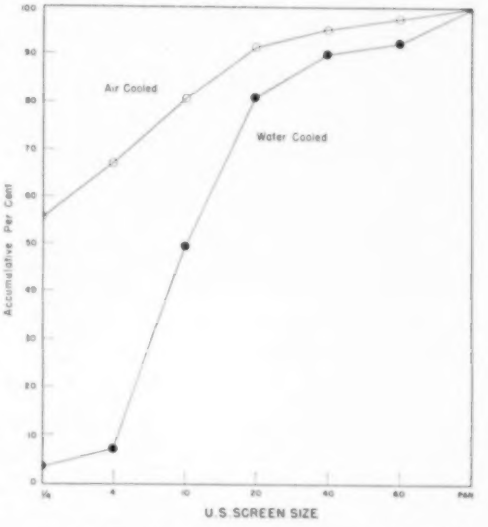

Fig. 16—Tumbler test results, water and air-cooled sinter.

4. Magnetite grains in water-cooled samples are not as coarse as those in air-cooled samples.

Summary

Investigational work has led to the development of two physical tests for sinter. These have been found to be valuable tools in experimental work.

Both fuel and moisture contents of sinter mixes are important variables requiring close control for

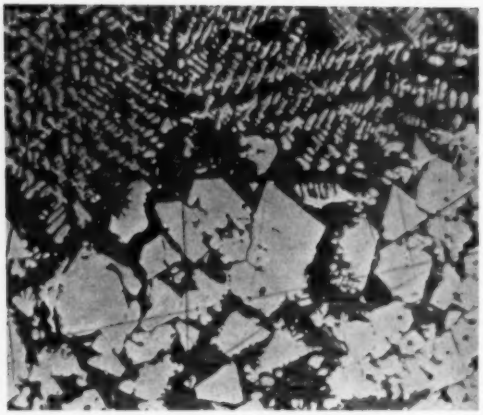

Fig. 17—Hard magnetite sinter. White areas are magnetite, light gray areas are slag. (Water cooled.) X100.

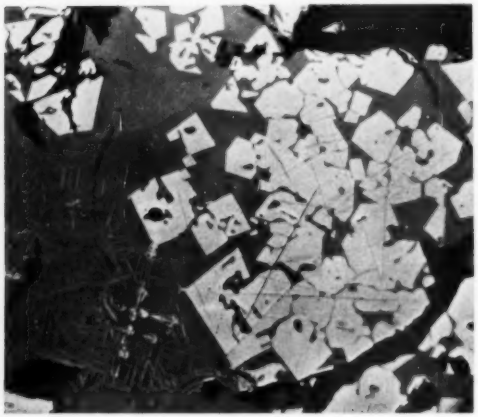

Fig. 18—Soft magnetite sinter. White areas are magnetite, light gray areas are slag. (Water cooled.) X100.

uniform operation and a uniform product. It is our observation that sufficiently accurate controls for these variables are not now available to the operators.

Reducibility tests show softer sinters to be less difficult to reduce than hard sinters, but blast furnace operators desire the hard sinters to avoid breakup in handling. Along this line, it is possible to decrease the amount of breakup if the sinter is air cooled.

Iron ores divided into three classes according to particle size distribution have been used as a basis for the work. The coarse group of ores sinter at a much faster rate than those of the intermediate group. In order to speed up the sintering rate of our intermediate group, it is necessary to improve the permeability of the mix layer, as distinguished from the three other layers present during sintering. Additions of prepared solids to the mix can aid in this respect. When prepared agglomerates of the ore itself are added, difficulty in maintaining good permeability throughout the cycle is encountered as well as the further resistance at times of the agglomerates to fusing completely. Sized returns appear highly desirable, but the overall economics of their plant application has not been determined. The partial use of sized lightweight fuels increases the sintering rate considerably. However, because of high transportation costs of bulky material, application appears limited to plants located close to sources of supply.

As mentioned previously, we have as yet carried out little work on the fine group of ores. It seems to be the opinion in some quarters that sintering of fine material will not meet with undue difficulties, and the operation of the old Mesabi Iron Co. in Minnesota is cited as an example. We have no doubt that fine ores can be sintered, but there appears to be the important problem of sintering at an acceptable production rate. Examination of the literature[6, 7, 8] shows the above plant to have operated a 42-in. wide Dwight-Lloyd machine with a 6-in. bed at a speed of only 48 in. per min.

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Sampling the Mining News

Discovery of a hitherto unsuspected extension of the Vermilion range iron formation in Northern St. Louis County, Minn., has been announced by the director of the State's geological survey, Dr. George M. Schwartz. USGS aeromagnetic maps show that the Vermilion range curves southeast from the Tower-Soudan area into the wilderness between Embarrass and Bear Head Lake. Dr. Schwartz believes that this new information justifies considerable exploration and drilling for iron ore in an area of about 20 sq miles southeast of Soudan. The aerial maps were a result of a three year USGS-Minnesota Geological Survey aerial search for iron formations in a 30,000 sq mile area in northern Minnesota.

The increased demand for copper, lead, zinc, and manganese has lead to increased mining activity in southern and southwest New Mexico. Numerous small producers which suspended operations after the war are again going into production. The Merrimac mine in the Organ district has been leased, and AS&R's new mill at Deming is taking its shipments of lead-zinc ore. In Socorro County the Hurlow Mining & Milling Co. and the Portales Mining Co. have begun operations shipping to the El Paso smelter. Numerous leases are also opening up in Grant County, developing promising copper, lead, and zinc prospects.

About 1900 employees of Kennecott's Chino Mines Div. were recently granted a 10 pct hourly wage increase, retroactive to Oct. 15. The increase brought workers' base pay to just over $72 a week for 48 hr. The wage boost, which was prompted by an increase granted Oct. 8 by the Phelps Dodge Corp., amounts to $2.50 a week for each employee, and blankets all employees, represented by 6 different unions.

A $200,000 processing plant is proposed for preparation of powdered mica from dumps of Colonial Mine Co., Moscow, Idaho. The Latah County, Idaho, deposits also contain beryl, and there is interest in the beryllium content of the material. The processing method to be used is reported to be a new one, but whether the plant will turn out a finished product or merely treat the material and ship it elsewhere for finishing is not known.

A four-year contract has been awarded Domestic Manganese Production Co., for production and processing manganese ore for the Government stockpile. Deposits are in the Butte and Phillipsburg, Mont., districts. About 100 men are expected to be added to employment rolls at the mill, and about 200 at mining properties.

New mining and milling ventures are top news in the Salmon, Idaho, area. Howe Sound Mining Co. is completing its Calera plant for cobalt refining; tungsten is being mined by Bradley Mining Co., along with gold, lead, and zinc; and other mining and milling ventures are in prospect. The Blackbird mine of Calera Mining Co., a Howe Sound subsidiary, has been under development for five years, and underground workings total 30,000 ft. A 600-ton concentrator is being completed.

A new deep shaft development is proposed by Silver Buckle Mining Co., and 2800 acres are either owned or under option. If the project goes through it will be a joint venture of several companies involving about $1.5 million expenditures before any returns are expected. The tentative location of the 3500-ft shaft is reported about a mile from Wallace, Idaho.

Gas from lignite is expected to ultimately yield a power source big enough to operate American industry for 400 years. Resources total about 900 million tons, and a $750,000 lignite laboratory is being completed in North Dakota by Bureau of Mines. Lignite is converted to gas by a special technique, and the gas has a variety of end uses. It can be used as a gas for burning, and it can be converted to gasoline and diesel fuel. It can also be used to make alcohol, ammonia, waxes, dyes, and coal by-products.

The Atomic Energy Commission has lowered the minimum grade of uranium development ore that will be accepted from producers in the Marysville, Utah, area, as an incentive to uranium prospecting and development there. The minimum grade is now 1/10 of 1 pct uranium oxide, instead of the 2/10 of 1 pct heretofore acceptable. Development allowances for these low-grade ores will be set so that payment, including development allowance, for ores containing 0.10 pct to 0.20 pct U308 will be $2.50 per lb of contained U308. This will apply only to ores extracted during the prospecting and development stages.

Uranium has been discovered 10 miles east of Hot Springs, Ark., by University of Arkansas research parties. The find was in the Potash-Sulphur Springs area, not far from Magnet Cove. Ore samples contained as high as 1 pct of uranium. Extensive drilling will be required to determine the extent and proportion of high grade ore and the presence of other minerals, according to W. W. Grigorieff, director of the university's institute of science and technology.

Canada

Unwatering of the Harrison-Hibbert Mines North Cobalt silver property has started, the work being done through a financing agreement with Macfie Red Lake Mines. The property was last active in 1948, when a 160-ft shaft was sunk and drifting and cross-cutting carried out. Some high grade silver and cobalt was revealed. About 1012 lb of high grade were hand sorted to yield a grade of 3800 oz of silver per ton.

A fourth drill has recently been added to the Bethlehem Steel Co.'s diamond drill exploration of magnetic anomalies in search for magnetite deposits near Marmora, in Southeastern Ontario. Drilling started last June, and operated most of the summer. Bethlehem geologist W. G. Wahl is in charge of the project.

Abroad

Thailand (Siam) is facing a shortage of tin deposits and miners are demanding government action in prospecting to find new sources of the metal. Thailand tin mines are working at about two-thirds of prewar capacity, but efforts are being made to increase their current production of 14,000 tons per year.

In Berlin, it was reported that a rich uranium vein has been found in East Germany. Wismuth A. G., Soviet-controlled organization, is said to have found the vein between Lictentanne, near Zwatau, and the Roter Berg near Werdau, on the Czech frontier.

It has also been claimed that Russia has imported 500 Chinese to mine newly discovered uranium deposits in Polish-occupied Lower Silesia. A Berlin newspaper said that 20,000 Chinese are expected to be settled in Silesia by the end of the year as cheap labor for Polish-operated coal mines and lumber camps.

Portable Roof Bolt Pull Tester

Answering the need for a portable test unit to actually determine roof bolt anchorage, the Continental Gin Co.'s Model BT 252 tester is also an aid in the search for the most effective length and

type of bolt for anchorage in any strata. Existing, normally installed bolts are tested. A set of adaptors for attachment to $\frac{5}{8}$, $\frac{7}{8}$, and 1 in. diam bolts is supplied with the unit. The unit consists of a hydraulic ram, hydraulic pump, hydraulic gage with equalizer and hydraulic hose—all conservatively rated for 25-ton pull. A reinforced column supports ram thrust, and a high strength take up nut and adaptors apply ram pull. A set of safety chains is supplied for protection in case of bolt failure under test. **Circle No. 1**

Glove Salvage Cuts Costs

A new work glove reconditioning service is now being offered on a national basis. Service includes complete renovation, sanitizing and packaging of all worn-out fabric and leather gloves and protective clothing. It has been discovered that most gloves which were thrown

away until now can be re-used. Reconditioning costs only a fraction of new glove cost, so that material savings are claimed. Salvage, on a study involving over 1 million gloves of every type from many industries, averaged 83.9 pct. U.S. Industrial Glove Corp. offers convenient transportation, at no additional expense, in industrial areas. A free trial plan is available. **Circle No. 2**

New Coal Shaker Cam Grease

A difficult lubrication problem which has long beset the coal preparation field is said to have been solved by Gulf Cam Grease, designed especially for shaker screen cams. The product is already in extensive use at several anthracite mines. Shaker screen cams and eccentrics often develop temperatures of 250°F and higher. The new grease will cling to bearing surfaces at such operating temperatures. It is easily applied by compression grease cup, pressure gun, or automatic systems. **Circle No. 3**

Silver Plating Powder

A new method of silver plating high-amperage electrical connections helps improve continuity of electrical service, and reduces maintenance time and costs. Cool-Amp silver plating powder requires only

wire brush, clean cloth and water for application, and provides cool maximum conductivity for conducts. A pound will silver plate 6000 sq in., and sells for $11.50. Now being used by Kennecott Copper Corp., Ohio Edison Co., Florida Power & Light Co., and other organizations. **Circle No. 4**

New Battery-Powered Trammer

A $1\frac{1}{2}$-ton battery-powered trammer for haulage work has been developed by the General Electric Co. for use in metal mines with restricted clearances. Available in any track gage between 18 and 24 in., the unit is $71\frac{1}{2}$ in. long over bumpers, $35\frac{5}{16}$ in. wide and weighs 3000 lb with battery. With a standard battery it is $38\frac{1}{8}$ in. high, and 6 in. higher with a high-type battery. Rated drawbar pull is 400 lb, with a maximum drawbar pull on level tangent of 750 lb. It travels at 7 mph, or 3 mph with rated drawbar pull. The locomotive has a folding cab which permits loading in limited mine hoist cages. **Circle No. 5**

Carbide Bit Grinding Gage

Here's a handy gage for use in reconditioning tungsten-carbide bits, showing when to grind them, and the proper angle to grind a dull bit. Rock Bit Sales & Service Co. sells it. **Circle No. 6**

Lighter, Lower Air Compressor

Worthington Pump & Machinery Corp. is now offering a 105-lb portable air compressor, built around the standard Worthington Blue Brute compressor, and powered by either diesel or gasoline engine. New features claimed are: retractable third wheel, underslung spring-mounted undercarriage with 15-in. tires, new style radiator for better operation at high temperatures and altitudes, a new carburetor with fixed jets for economy, and unrestricted full length tool boxes. The compressor weighs 300 lb less than previous models. **Circle No. 7**

Portable Mine Lighting System

A low-voltage portable lighting system for mining and construction work has been announced by Joy Mfg. Co. The compressed-air driven Joy-Lite develops an output of 250 to 280 watts, enough to operate 4 high-powered sealed-beam floodlights. The generator is mounted in a steel box equipped with a carrying handle. Lamps have tripod swivel bases which can be used as column clamps, and hooks serving as hangers. Unit weighs 48 lb. **Circle No. 8**

All-Service Rubber Hose

A new all-service hose called Servall is being offered by the Hewitt Rubber Div. of Hewitt Robins, Inc. It can be used, (according to recommended pressures) for air, hot and cold water, can handle mild chemicals, grease, paint, and can be used for low-pressure chemical spray service. The tube is of synthetic rubber, the carcass of rayon imbedded in synthetic rubber, and the cover is weather and abrasion resistant. Available in 10 different ID's, weighing from 12 to 69 lb per 100 ft. **Circle No. 9**

Free Literature

10) SPEED CHANGERS: available in three different series for various applications, miniature speed changers are described in brochure available from *Metron Instrument Co.* These small lightweight gear units are the solution to fixed ratio gearing problems in instruments, servomechanisms, controls, regulators, etc. Any of hundreds of standard ratios from 10/9 to 531,440/1 are assembled from stock parts for immediate shipment.

11) SILICONE PRODUCTS: A reference guide to these products manufactured by *Dow Corning Corp.* lists fluids, compounds, greases, and resins and varnishes, and silastic. Booklet numbers are given for more detailed information on specific subjects.

12) SPEED REDUCERS: Bulletin 130 describes *Cleveland Worm & Gear Co.*'s type AT and RT worm gear speed reducers. Several engineering improvements are listed such as face type oil seals used in place of the older style packing gland on type AT units. New higher hp rating tables are easier to work with for they show output speeds and the more frequently used 1750 rpm ratings. Cross-sectional diagrams and dimensions are shown on the various sizes and types.

13) SELF-PRIMING PUMPS: New catalog bulletin including descriptions of 3 in. 15 M self-priming centrifugal pump as well as the 7M and 10M sizes may be obtained from *Rice Pump & Machine Co.* These sizes are also available with pulleys for belt drives and flexible couplings for direct-connected electric motor drives.

14) PLANT DESIGN: Consultation and development, design of mill or plant, and cost estimates, are just a few of the services offered by *O. W. Walvoord Co.* Booklet illustrates mills now in operation in Colorado, Nevada, and Utah and notes companies that use this service both for consultation and actual construction.

15) THERMOCOUPLES: Catalog 5 issued by *Arklay S. Richards Co., Inc.* illustrates thermocouples, protection tubes, thermocouple wire, lead wires, and insulators for all makes of pyrometers. Gage, ID and OD size, material or composition, maximum temperatures, and insulations are all listed in tables.

16) HOPPER CAR UNLOADER: These unloaders are used for all materials as gravel, coal, coke, clay, foundry sand, earth, etc. Many advantages of these Barber-Greene unloaders are: positive non-slip drive, one-man operation, 25 mph road speed, operates in pit or above rails. Model 358 is specified.

17) REGULATORS: A new 4-page folder may be obtained from *Spence Engineering Co., Inc.* which gives useful information on temperature regulators, pressure regulators, and desuperheaters. A complete explanation of the principles of pilot-operated regulating valves is given in this folder in addition to data and illustrations of 27 Spence pilots.

18) CONVEYOR BELT: Homocord conveyor belts are illustrated and described in bulletin available from *Raybestos-Manhattan, Inc.* Unlike regular cord type belts, these belts hold metal fasteners and can also be spliced and vulcanized in the field. Recommended types for various applications are given.

19) APRON FEEDERS: The basic feeder unit is furnished without sideboards to include one of the three standard drives: no speed reduction, for use with gear head motors; with two counter shafts, for use with conveyors; and with speed reduction unit, for use with crushers. Bulletin may be obtained from *Pioneer Engineering Works* illustrating these feeders which are sturdily built for pit, mine, and quarry operation.

20) CHARGING EQUIPMENT: Brochure available from *Electric Storage Battery Co.* tells the entire story of charging equipment for exide batteries, motive power service. The fundamental principles, automatic charging, automatic two-rate charging and charging equipment are all covered.

21) BLOCKS: Rope Master slusher blocks in medium, heavy, and super duty sizes and utility blocks are illustrated in pamphlet offered by *Paul E. Keeney Co.* A chart is contained to help in selecting the proper block. Sheave size, safe load lb, weight with swivel hook, and ultimate strength in lb are listed in the chart.

22) DRILL BIT: The Liddicoat bit is a one-use throw-away rock bit, made from specially selected forged and heat-treated steel. It is adaptable to practically all mining and rock-drilling operations. Its use is not only confined to metal mining, but is rapidly finding increasing application in a diversity of rock-drilling operations. *Calumet & Hecla Consolidated Copper Co.* has issued a circular containing full information on this detachable drill bit.

23) WOOD TREATMENT: Osmoplastic, is a scientific blending of recognized preservatives to extend the service life of wooden poles, butt ends of railroad and railway mine ties, etc. When any timer is treated with this composition the moisture becomes an ally rather than an enemy acting as a vehicle which carries the toxic salts in Osmoplastic far into the wood, thus protecting the wood against fungi and consequent decay. The entire story is told in booklet offered by *Osmose Wood Preserving Co. of America, Inc.*

24) IGNITRON SUBSTATIONS: When the working face has advanced, these portable Ignitron rectifier substations move forward on the regular mine tracks or are skidded to the new location. This keeps the dc voltage high at the face, and keeps the mining operation efficient. Circular issued by *Westinghouse Electric Corp.* presents information on the operating features of the substations.

Mining Engineering
29 West 39th St.
New York 18, N. Y.

December

Please send me { More Information ☐ / Free Literature ☐ / Price Data ☐ } on items indicated.

1	2	3	4	5	6	7	8	9	10
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Name ..... Title .....

Company .....

Street .....

City and Zone ..... State .....

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aime NEWS

Tentative Annual Meeting Program

The annual meeting of the American Institute of Mining and Metallurgical Engineers will be held at the Jefferson and Statler Hotels, St. Louis, Feb. 19 to 22, with the former as official headquarters. The Welcoming Luncheon, Smoker and Student Forum on Monday night, the Informal Dance, Tuesday night, and the Annual Banquet and President's Reception, Wednesday night, will be at the Jefferson. The Metals Branch and Petroleum Branch dinners on Tuesday night will be at the Statler. The Mining Branch, except for three sessions, at the Statler, will present its technical program at the Jefferson and the Metals and Petroleum Branches will hold their sessions at the Statler.

The Council of Section Delegates will meet on Saturday and the Board of Directors will convene in general session on Sunday afternoon at the Jefferson. The annual business meeting of the Institute will be on Tuesday afternoon. The Mineral Industry Education Div. will hold its usual Sunday afternoon and evening sessions.

Whereas the technical program for Monday through Thursday still is in a state of flux and the meeting schedules shown here are tentative, there is some fixation in that rooms of appropriate seating capacities are assigned to sessions of anticipated attendances.

Mining Branch

Monday Morning—A joint session of the Mining and Geology Subdivisions to discuss geology and the choice of mining methods; the Coal Div. will devote the morning to the coal depreciation problem, and the Mineral Industry Education Div. will hold its final technical session for the meeting. The Industrial Minerals Div. will hold the first of two sessions on Canadian industrial minerals. Not yet jelled, nevertheless a possibility, is a session of the Industrial Minerals Div. to take the strain off its heavy schedule Monday afternoon through Thursday morning.

Monday Afternoon—Mineral Economics Div., foreign minerals; joint session Mining and Geology Subdivisions; joint session, Geology Subdivision and Society of Economic Geologists, symposium on ground water geology; Coal Div., depreciation; Minerals Beneficiation Div., materials handling; Industrial Minerals Div., general; Industrial Minerals Div., Canadian minerals.

Tuesday Morning—Mineral Economic Div. and Geology Subdivision, exploration costs; Mining Subdivision; Coal Div., general session; Industrial Minerals Div., dimension stone; Geology Subdivision and SEG, surface and near-surface manifestations of ore bodies; Industrial Minerals Div., clay. A joint session of Minerals Beneficiation and Extractive Metallurgy Divisions on solution and precipitation has been mentioned as a possibility.

Tuesday Afternoon—Mining Subdivision; Mineral Economics Div., taxation, tariff and subsidies; Coal Div., general session; Minerals Beneficiation, concentration; Geology Subdivision and SEG; Industrial Minerals Div., Geology Subdivision and SEG, clay and bauxite: genesis, occurrence, exploration, and laboratory studies; Industrial Minerals Div., aggregates; Industrial Minerals Div. and Mining Subdivision, mining methods.

Wednesday Morning — Industrial Minerals, Geology Subdivision and SEG, clay morphology and genesis; Mining Subdivision; Coal Div., preparation; Coal Div. and SEG, geology; Industrial Minerals Div., general session; Geophysics Subdivision.

Wednesday Afternoon—Industrial Minerals Div., Geology Subdivision and SEG, clay, clean-up; Mining Subdivision; Mineral Economics and Industrial Minerals Divisions, cost accounting in mining; Coal Div. and SEG; Geophysics. Geology Subdivision, general session.

Thursday Morning—Mineral Economics and Industrial Minerals Divisions, economics of industrial minerals; Industrial Minerals Division, general session. Minerals Beneficiation Div., symposium.

Thursday Afternoon—Minerals Beneficiation Div., concentration and clean-up session.

LUNCHEONS

Luncheons of Mining Branch Divisions scheduled so far: **Tuesday,** Mining, Geology, and Geophysics Div.; Minerals Economics Div., luncheon and business meeting; Mineral Industry Education Div.; **Wednesday,** Industrial Minerals Div.; **Thursday,** Minerals Beneficiation Div.

Annual Meeting Field Trips

Reservations will be limited to plant and transportation facilities and must be made immediately to assure a place on the trip. All trips will be on Friday, Feb. 23, and will take the entire day, except Trip No. 2, which will leave St. Louis 11:20 p.m. Thursday and return to St. Louis, Saturday, 7:25 a.m.

Trip No. 1 Lead Belt, Southeastern Missouri; mining, geology, and ore dressing; Bonne Terre and Flat River, Mo.

Trip No. 2 Tri-State Zinc District, Missouri, Kansas, Oklahoma; mining, geology, ore dressing, and lead smelting; Joplin, Mo., and Picher, Okla.

Trip No. 3 Zinc Smelters, horizontal retort and electrolytic smelting; Fairmont and Monsanto, Ill.

Trip No. 4 Midwest-Radiant Coal Mine and Washing Plant, Millstadt, Ill., and coking unit of Koppers Co., Granite City, Ill.

Trip No. 5 U. S. Bureau of Mines, synthetic fuels (oil from coal); Louisiana, Missouri.

Trip No. 6 Pittsburgh Plate Glass Co., mine and plant (plate and safety glass); Crystal City, Mo.

Committees for February 1951 Annual Meeting Revised

General Committee:

Howard I. Young, *Chairman and Fred J. Meek, Secretary.*

H. E. Butters, E. L. Clark, Arthur B. Cleaves, L. P. Davidson, Albert J. Frank, Herbert A. Franke, Jules C. George, Neal Ham, Norman Hinchey, John D. Kerr, K. A. Phillips, Nat L. Shepard, Carl G. Stifel, Carl Tolman, Curtis L. Wilson.

Finance Committee:

Carl G. Stifel, *Chairman*

Jules C. George, *Vice-Chairman*

F. W. Green, K. A. Spencer, O. W. Bilharz, Joseph Desloge, G. S. Jenkins, B. F. Murphy, Elmer Isern.

Reception Committee:

Carl Tolman, *Chairman*

Curtis L. Wilson, *Vice-Chairman*

Fr. J. B. Macelwane, M. M. Leighton, Victor Rakowsky, Richard W. Love, H. K. Hanley, J. H. Steinmetz, D. F. Walsh, D. R. Schooler, Harold A. Krueger, Elmer Isern, H. L. Walker, Ernest Ohrie, J. W. MacDonald, W. T. Isbell, Richard Bullock, E. L. Hockman, William D. Beck, M. M. Fine.

Banquet Committee:

John D. Kerr, *Chairman.*

Herbert R. Hanley, Arthur Bevan, Oliver R. Grawe, N. A. Stockett, Alfred H. Bell, Fr. J. B. Macelwane, J. D. Forrester.

Informal Dance Committee:

K. A. Phillips, *Chairman*

H. E. Butters, *Vice-Chairman*

C. DeWitt Smith, G. C. Bartholomees, Verne Pulsifer, Mrs. G. L. Spencer, Jr., Mrs. Carl Stifel, Mrs. H. A. Neustaedter.

Dinner-Smoker Committee:

Herbert A. Franke, *Chairman*

Neal Ham, *Vice-Chairman*

Paul Hamilton, G. S. Jenkins, Clyde S. Smith, Bernard F. Desloge, Gill Montgomery, H. LeRoy Schanen.

Welcome Luncheon Committee:

L. P. Davidson, *Chairman*

A. W. Schlechten, L. A. Painter, C. V. Cameron, H. J. Tschirner, O. W. Bilharz.

Hotel Committee:

Nat L. Shepard, *Chairman*

Albert J. Frank, *Vice-Chairman*

G. H. Cady, F. S. Elfred, J. Marshall Thompson, C. W. Ambler, Jr.

Publicity Committee:

Arthur B. Cleaves, *Chairman*

H. E. Butters, *Vice-Chairman*

John S. Brown, O. M. Bishop, A. B. Campbell, Harold Underhill.

Field Trips Committee:

E. L. Clark, *Chairman*

Norman Hinchey, *Vice-Chairman*

H. A. Neustaedter, R. G. Knickerbocker, Frank H. Reed, S. S. Clarke, H. R. Hanley, Amzi G. Garrard, J. E. Lamar, Jack James, W. D. Keller, John S. Brown, Arthur Mallams, H. E. Gross.

Student Activities Committee:

Curtis L. Wilson, *Chairman*

Albert J. Frank, *Vice-Chairman*

A. F. Frederickson, D. R. Schooler, Charles O. Frush, D. F. Walsh, W. R. Chelsey.

WOMAN'S AUXILIARY

Mrs. Howard I. Young, *General Chairman*

Ladies' Entertainment Committee:

Mrs. Carl Tolman, *Chairman*

Mrs. Carl Stifel, *Vice-Chairman*

Mrs. L. P. Davidson, Mrs. Kenneth Phillips, Mrs. N. L. Shepard, Mrs. Fred J. Meek, Mrs. Jules C. George, Mrs. J. A. Rood, Mrs. Otis Banes, Mrs. C. W. Ambler, Mrs. George Bartholomees, Mrs. Floyd Belleau, Mrs. Herbert A. Franke, Mrs. George Spencer, Jr., Mrs. H. A. Neustaedter, Mrs. W. W. Weigel, Mrs. Violet Wilson, Mrs. Curtis L. Wilson, Mrs. R. B. Fisher, Mrs. H. R. Hanley, Mrs. J. D. Forrester, Mrs. Edward Clark, Mrs. R. L. Raymond, Mrs. R. G. Knickerbocker, Mrs. A. W. Schlechten, Mrs. Adolph Legsdin, Mrs. Carl Cotterill, Mrs. W. T. Isbell.

Conditions Under Which 1951 Publications are Available to Members

Pursuant to Article X of the By-laws of the AIME, the following information is hereby given as to the "conditions, prices, and terms under which the various classes of members, and Student Associates, severally, shall be privileged to obtain publications of the Institute during the ensuing year."

Publications authorized for 1951 publication include the following: Mining Engineering, published monthly, containing material, including technical papers, of interest to those engaged in exploration, mining geology and geophysics, and metal, nonmetallic, and coal mining and beneficiation, and fuel technology. The Journal of Metals, published monthly, containing material, including technical papers, of interest to those engaged in nonferrous smelting and refining, iron and steel, and physicial metallurgy. The Journal of Petroleum Technology, published monthly in Dallas, containing material, including technical papers of interest to those engaged in petroleum production.

Current copies of any one of the above journals will be supplied to all members in good standing without further charge. (A member ceases to be in good standing if current dues are not paid by April 1.) If more than one of the monthly journals is requested, $4 extra will be charged for an annual subscription, or 75¢ for single copies. Higher prices may be charged for special issues. The nonmember subscription price for each journal is $8 in the Americas; foreign, $9. Student Associates who pay dues of $4.50 will be entitled to the same privileges for all publications as members except as noted below; those who pay $2 will not receive the journals but otherwise have the same privileges.

Three volumes of "Transactions" are authorized for 1951 publication, as follows: No. 187, Mining Branch; No. 188, Metals Branch; and No. 189, Petroleum Branch. These volumes will be available to members at $3.50 each for a first copy if paid for in advance with dues; otherwise at the nonmember rate of $7 less 30 pct. Nonmembers $7 in the United States; foreign $7.50.

Special volumes now planned for publication in 1951 include the following: (1) Open Hearth Proceedings. Price to AIME members $7; nonmembers $10. (2) Blast Furnace, Coke Oven, and Raw Materials Proceedings. AIME members $7; nonmembers $10. (3) Electric Furnace Steel Proceedings. AIME members $7; nonmembers $10. (4) Symposium on Tube Production Practice. Price to be set. (5) Basic Open Hearth Steelmaking, second edition. Price to be set. (6) Statistics of Oil and Gas Development and Production, covering data for year 1949. AIME members $3; nonmembers $6. (7) Petroleum Conservation, by Stuart Buckley. Price to be determined. (8) Symposium on Solidification of Metals and Alloys. Price to be set.

If dues are paid subsequent to Jan. 31, back issues of Institute publications will be supplied only if adequate stocks are on hand. A member may not receive a volume of "Transactions" or a special volume in lieu of a monthly journal, free of charge on membership. Members in arrears for dues are not entitled to special members' prices for publications.

Rocky Mountain Members may have their choice of an annual subscription to one of the monthly journals on request.

Annual Business Meeting

Notice is hereby given that the Annual Business Meeting of the AIME will be held at the Hotel Jefferson, St. Louis, at 4 p.m. Tuesday, Feb. 20, 1951. The newly elected Directors and officers assume their duties at this meeting. A report of the financial condition of the Institute will be made, as well as reports of the principal officers and standing committees. All AIME members are invited to be present. Immediately afterward, at 5 p.m., the new Board will meet.

AIME Financial Position

Reviews of the financial position of the AIME in the current year, as made at recent meetings of the Directors, indicate that a deficit of some $20,000 may be expected. The budget at the beginning of the year had been precariously balanced at $477,500, but in some instances income has been less than expected and expenses greater. Chief drop from expected income has been in advertising in the three journals, which had been budgeted at $125,000 and now promises to be about $105,000. Expense for the journals may be slightly higher than expected, owing to increased printing and paper costs and to a policy of maintaining their high-standard with the hope of achieving their acceptance as desirable advertising media. The directory proved more expensive than anticipated. On the other hand, dues income has been higher than expected, and certain economies in administration and office practices have been attained. Even with the publication of the directory this year, an expense which was not borne in 1949, the deficit promises to be reduced by some $50,000.

Payment of 1951 Dues

Pursuant to Article II, Section 2, of the bylaws of the AIME, notice is hereby given that dues for the year 1951 are payable Jan. 1 as follows: Members and Associate Members, $20; Junior Members for the first six years of Junior Membership, $12; Junior Members, thereafter, $17; Student Associates (including an annual subscription to a monthly journal, $4.50; Student Associates (without Journal subscription), $2.00.

Mailing of dues bills began in September and will be completed in December. Prompt payment will assure uninterrupted receipt of the journal or journals desired in 1951. If, for any reason, a bill is not received within a reasonable time after the last mailing date, Dec. 31, headquarters should be notified.

Coming Events

Dec. 13, AIME, San Francisco Section, Reminiscence meeting, Engineers' Club, San Francisco.

Dec. 14, AIME, Ajo Subsection, Assembly Hall, Phelps Dodge Corp., Ajo, Ariz.

Dec. 26-31, American Assn. for Advancement of Science, annual meeting, Convention Hall and University of Pennsylvania Campus, Philadelphia.

Jan. 15-17, 1951, AIME, Minnesota Section, annual meeting. Mining symposium conducted by Center for Continuation Study, University of Minnesota.

Jan. 18-20, Society of Plastics Engineers, annual national technical conference, Hotel Statler, New York.

Feb. 18-22, AIME, annual meeting, Jefferson Hotel, St. Louis. **Metals** Branch session to be held at the Statler Hotel.

Mar. 5-9, ASTM, spring meeting and committee week, Cincinnati.

Mar. 13-16, National Assn. of Corrosion Engineers, conference and exhibition, Hotel Statler, New York.

Apr. 2-4, 1951, AIME, Open Hearth and Blast Furnace, Coke Oven and Raw Materials Conference, Iron and Steel Div., Statler Hotel, Cleveland.

Apr. 2-5, ASME, spring meeting, Atlanta-Biltmore, Atlanta.

Apr. 23-26, American Foundrymen's Society, annual convention, Buffalo, N. Y.

May 9-11, Engineering Institute of Canada, annual meeting, Mount Royal Hotel, Montreal.

May 14-17, American Mining Congress, coal convention and exposition, Public Auditorium, Cleveland.

Nominating Committee for 1952 AIME Officers

As provided in Article IX, Section 2, of the AIME by-laws, the names and addresses of the Nominating Committee for Institute officers for 1952 are given below. In each case, the name in parenthesis is that of the alternate, who is to serve only in the absence of the principal at the meeting of the Committee.

Members designated by the Council of Section Delegates: W. L. Zeigler, general manager, Pend Oreille Mines & Metals Co., Metaline Falls, Wash. (Wallace G. Woolf, Box 209, Kellogg, Idaho); Henry T. Mudd, 1206 Pacific Mutual Bldg., Los Angeles 14. (James A. Kavenaugh, 950 N. Highland Ave., Los Angeles 38); Albert Mendelsohn, president and general manager, Cananea Consolidated Copper Co., Cananea, Son., Mexico (P. D. I. Honeyman, vice-president and general manager, Inspiration Consolidated Coper Co., Inspiration, Ariz.); Herbert M. Cooley, assistant metallurgical engineer, Bethlehem Steel Co., Box 2171, Tulsa 2. (P. P. Manion Jr., Stanolind Oil & Gas Co., Stanolind Bldg., Tulsa); Paul R. Turnbull, manager drilling and production department, La Gloria Corp., P.O. Box 779, Corpus Christi (M. D. Cooper, National Coal Assn., 5430 Aylesboro Ave., Pittsburgh 17); Philip D. Wilson, Lehman Bros., 1 William St., New York (James Douglas, Phelps Dodge Corp., 40 Wall St., New York 5); R. Schuhmann Jr., associate professor of process metallurgy, MIT, 77 Massachusetts Ave., Cambridge 39 (H. Rush Spedden, MIT, Cambridge 39); Herbert R. Hanley, emeritus professor of metallurgy, Box 318, Rolla, Mo. (L. P. Davidson, Box 222, Monsanto, Ill.); Richard F. Miller, assistant to vice-president, research and technology, Carnegie-Illinois Steel Corp., 434 Fifth Ave., Pittsburgh 30 (John T. Ryan Jr., Mine Safety Appliances Co., 201 N. Braddock Ave., Pittsburgh 8); Harry M. St. John, superintendent, Department 5, Crane Co., 4100 S. Kedzie Ave., Chicago 5 (Rudolph Tietig Jr., engineer, A. J. Boynton & Co., 58 E. Washington St., Chicago 2); E. D. Dickerman, 2251 S. Cook St., Denver 10 (C. L. Barker, 308 Jasmine St., Denver).

Members designated by the Branch Councils: S. J. Swainson, director mineral dressing laboratory, American Cyanamid Co., 1937 W. Main St., Stamford, Conn. (Veleair C. Smith, Kanawha Valley Bldg., Charleston, W. Va.); J. Murray Riddell, head, department of mining engineering, Michigan College of Mining and Technology, Box 144, Houghton, Mich. (Richard W. Smith, 1615 H St., N.W., Washington 6, D. C.); Harold K. Work, director, research division, College of Engineering, New York University, University Heights, New York 53; Lloyd E. Elkins, production research director, Stanolind Oil & Gas Co., P.O. Box 591, Tulsa 2 (R. C. Earlougher, 319 E. 4th St., Tulsa 3).

Members designated by President McLaughlin:

Chairman: Clayton G. Ball, vice-president, Paul Weir Co., 20 N. Wacker Drive, Chicago 6. (Philip R. Bradley, Jr., president and general manager, Pacific Mining Co., 1022 Crocker Bldg., San Francisco.) John R. Suman, vice-president and director, Standard Oil Co. (N.J.) 30 Rockefeller Plaza, New York 20. (Howard C. Pyle, president, Continental Consol. Corp., 1562 Royal Blvd., Glendale 7.) P. C. Feddersen, general superintendent, Bunker Hill Smelter, Box 29, Kellogg, Idaho. (John D. Sullivan, Battelle Memorial Institute, 505 King Ave., Columbus.)

The Committee will welcome suggestions for the official slate for 1952. The following offices are to be filled: President for 1952, President-elect, (president for 1953), two Vice-Presidents, and five other Directors. The terms of the following Directors eligible for reelection expire in February 1952: L. E. Young, A. B. Kinzel, Philip Kraft, H. W. Johnson, William J. Coulter, Erle V. Daveler, James L. Head, and George P. Swift.

Drift of Things . . . . . as followed by *Edward H. Robie*

AIME members living outside of the United States frequently complain of the difficulty of securing American dollars with which to pay their dues. Efforts have been made to devise some means of easing the burden but so far without result. Deposits of foreign currency to the credit of the Institute cannot be utilized, for the expense of conducting the society has to be paid in American dollars.

As late as 1938, 25 pct of the membership resided in foreign lands. The exigencies of the last World War reduced this percentage to 16 pct in 1944, and it has since remained at between 15 and 17 pct. The current figure is 16 pct, and of this group of foreign members, 25 pct are in South America, 21 pct in Canada, 20 pct in Europe, and 9 pct in Mexico. It seems, therefore, that some 2600 AIME members living in foreign countries can still secure American dollars, so the problem cannot be too acute. Many, of course, maintain balances in banks in this country, particularly American engineers employed abroad.

In the last Directory, Toronto has been displaced as the city outside of the United States (we hesitate to call it a foreign city) with the largest number of AIME members. In the 1948 Directory the 1-2-3 rank was Toronto, London, and Caracas. Caracas now takes the lead, with just 100 on the Institute rolls, thanks to the oil and mining activity in Venezuela.

Utilization of Engineers

In the last World War the special talents of engineers, like those of many other groups of professional men, were to a large extent wasted. Further their training in the colleges was interrupted, so that the postwar supply was inadequate. Only in the last year has the supply equaled the demand. Members of the national engineering societies have often suggested that some action should be taken looking to a better utilization of their talents in future emergencies. They are not trying to evade service to their country, although few people look forward to taking up a gun and going out to shoot somebody, but they feel that it is to the country's interest as well as their own to serve where they can to the best advantage.

With the draft boards again picking up engineers for nonengineering military service, the problem has again become acute. Engineers Joint Council, representing the AIME, ASME, AIEE, ASCE, and AIChE, has therefore set up an Engineering Manpower Commission, in which the American Society for Engineering Education is also represented. Its function is to develop "policies and procedures designed to secure the most effective use of engineering skills and experience in industry and government (civilian and military) during the emergency and to take necessary steps within the scope of EJC to put such policies and procedures into practice." Three AIME representatives, one for each Branch of Institute activities, are acting on this Commission: George B. Corless, of the Standard Oil Co. (N.J.); M. W. Lightner, Carnegie-Illinois Steel Corp.; and H. J. O'Carroll, Kennecott Copper Corp.

The Commissions held an organizational meeting in New York on October 12, and two or three subsequent meetings have also been held or planned. Similar groups have been working through the American Chemical Society, the American Institute of Physics, and the National Research Council, and study of the general problem of the best utilization of professional men has also been undertaken by six committees advisory to General Hershey, director of Selective Service. These groups are reporting to the National Security Resources Board.

One is likely to underestimate the complexities of the problem, and perhaps to overlook the political angles. Almost every skilled laborer thinks he can serve his country better than by carrying a gun, and why should any white collar man be deferred. Military service by practically every physically fit man was fundamental in the idea of the Selective Service Act. Even if the idea be accepted that outstanding engineers and scientists be treated differently from others, the problem remains of registering them, finding the proper thing for them to do, and deciding on a cutoff point.

The Commission has made no final report, as this is written, but has agreed that fundamentally the long-range problem is twofold: (1) maintaining an adequate supply of personnel of the necessary abilities, and (2) assigning this personnel to the maximum advantage. The programs to be recommended must also take account of what is desirable on a large-scale or full mobilization basis, on a partial mobilization basis, with medium military forces, and lastly on the present 3 million-man military force basis.

To maintain an adequate supply of engineers it is felt that deferment from military service should be granted to those wishing to enter engineering schools who show exceptional promise, and that definite percentages of this group be advanced each year to the sophomore, junior, and senior classes. Eighty, 85, and 90 pct, respectively, have been suggested, so that 61 pct of a well-selected freshman group would graduate. Fixed survival percentages would prevent any complaints of academic grade juggling.

For all-out mobilization, registration of all scientific and technical personnel, both men and women, between the ages of 18 and 65 has been proposed. Registrants would include those with bachelor or higher degrees in engineering, physics, and chemistry; those enrolled in training programs; those employed in these fields without regard to education; and perhaps highly skilled technicians. Selective Service would handle the registration. The group thus registered, after proper screening, would constitute a "National Technical Reserve" membership in which would be comparable to membership in the armed forces or in the military reserve. Some badge, or other indication of membership, would be desirable because of social pressures.

A "National Technical Manpower Board", with regional and specialized committees, would be set up to assign duties to those registered, including assignment to and withdrawal from the military forces.

For partial mobilization, it is proposed that the above plan be adopted on a skeletonized basis, carrying assignments by the Board only to the extent necessary.

Admittedly, all this should have been done some time ago. But better late than never, and it is hoped that some action may result, either immediately or after any required legislation can be enacted, which will prevent some of the waste of technologic and professional manpower that has until now existed in this country in times of military emergency.

AIME Personals

Horace Y. Bassett, mining engineer, has been elected to the Board of directors of Calumet & Hecla Consolidated Copper Co. Mr. Bassett, a vice-president of the company since 1948, has been general manager of the Wolverine Tube Div., Detroit.

Rolland L. Blake has been awarded a research fellowship by the Federal Bureau of Mines in a cooperative agreement with the University of Minnesota. He will work toward a master's degree in geological engineering.

Robert Crist has been promoted to assistant superintendent of the M. A. Hanna Co.'s Cooley district iron mines on the Mesabi iron range.

Charles D. Clarke has accepted a position as mine superintendent with the Cerro de Pasco Corp., Peru.

Arthur Blake Caldwell is now assistant mining engineer with the U. S. Gypsum Co., Farnams, Mass.

Martial P. Corriveau recently completed the requirements for a master's degree at M.I.T. and is now an associate research professor in the department of mining engineering at Virginia Polytechnic Institute at Blacksburg, Va. He will conduct investigations in minerals separation.

Charles E. Dworjack-Graham is working with the Orinoco Mining Co. as a chief mechanical engineer at the construction of the future harbor of Puerto Caroni, Ciudad Bolivar, Venezuela.

George E. Erdman has accepted employment with the Susquehanna Collieries Div., M. A. Hanna Co., Nanticoke, Pa. upon graduation from Lehigh University. He is presently employed as a special mining engineer at the company's Glen Lyon, Pa. property.

Martial P. Corriveau

John A. Ferris, mechanical engineer with the Oliver Iron Mining Co., Duluth, was recently recalled to service in the armed forces.

Arthur R. Greenlee is now a geologist for the Bureau of Reclamation, Dept. of Operations & Development, Bismarck, N. Dak. His work is in the fields of glacial geology, stratigraphy, and hydrology.

F. J. Haller, superintendent of the Mather iron mine, has been appointed manager of Michigan iron mines of the Cleveland-Cliffs Iron Co. **James S. Westwater**, superintendent of the Mather "B" mine, becomes superintendent of the Mather mine.

Walter B. Johnson is now a metallurgist with the U. S. Bureau of Mines, Hydro-metallurgy & Ore Dressing Branch, Reno. He recently graduated from the Mackay School of Mines, University of Nevada.

H. F. Keller has returned to the U. S. from ten years in Bolivia with Patino Mines & Enterprises where he was employed as general manager of Bolivian Tin & Tungsten Mines Corp. Prior to going to Bolivia, Mr. Keller spent 15 years in Mexico at three different mining properties. Because of the political situation in Bolivia, Mr. Keller is trying to locate a position in the United States or Central America.

L. N. Kailasam, assistant geophysicist, Geological Survey of India, has joined the St. Louis University for advanced studies in geophysics.

H. E. La Tendresse is now an attaché at the U. S. Embassy, Pretoria, South Africa.

Abe P. Morris, formerly pit superintendent of Chino Mines Div., Kennecott Copper Corp., Santa Rita, N. Mex., has been named assistant to the general manager of Kennecott operations at Ray, Ariz.

Keith Edward Mantell has accepted an appointment as standards engineer in the Nchangar Consolidated Copper Mines, Ltd., Chingola, N. Rhodesia.

Cecil B. McGavock, Jr., of the Corps of Engineers has been transferred from the Baltimore District to the North Atlantic Div., New York City, where he is division geologist. He was formerly connected with the Seattle District of the Corps and the Tennessee Valley Authority.

Henry Joseph Petrie, New York City, has been appointed instructor in mining at Lafayette College, in the Mining & Metallurgical Engineering Dept. For the past two years he was an estimating and applications engineer with the Worthington Pump & Machinery Corp., Harrison, N. J.

J. Mauresse Powelson, consulting geologist, Haileybury, Ont., recently returned from a 3-month examination and evaluation of the silver-lead-zinc deposits and prospects in the Keno-Galena Hills area of the Mayo mining district in the Yukon.

A. V. Quine is now superintendent of zinc-lead property for the Goldfield Consolidated Mines Co., Northport, Wash.

Frank C. Roberts is now a mining engineer for the Climax Molybdenum Co., Golden, Colo.

Walter D. Roberts has joined the Coplay Cement Mfg. Co., Coplay, Pa. as assistant mining and field geologist engaged in the mining of limestone for use in the cement industry.

Hector A. Rochin has been employed by the Climax Molybdenum Co., Climax, Colo. since graduation from the Texas College of Mines.

J. Mauresse Powelson

Edward H. Rott has left for Caracas, Venezuela to act as regional manager for the Joy Mfg. Co., covering Venezuela, the Guianas, and the Caribbean area.

John F. Shaw is now a mining engineer with the U. S. Bureau of Mines, Denver, Colo.

Vernon E. Scheid has been appointed dean of the newly established College of Mining at the Mackay School of Mines, University of Nevada, Reno, consisting of the departments of geology, metallurgy, and mining and the public service State Bureau of Mines and State Analytical Laboratory. He will assume his duties on February 1, 1951. The name of Mackay School of Mines will be retained. **Jay A. Carpenter** will continue as director of the School until his retirement on July 1, 1951.

John F. Schultze was transferred from the Leadville Colo. Milling Unit of the American Smelting & Refining Co. to the Shorey Mill of the Northern Peru Mining & Smelting Co., Trujillo, Peru.

John Thomas Shimmin, Jr. has accepted a position as junior engineer with the Ray Mines Div., Kennecott Copper Corp., Hayden, Ariz. He was formerly second assistant chemist for the Utah Copper Div. of the same company.

Charles E. Stott, deputy director of the Marshall Plan's Strategic Materials Div., has been made director. **C. H. Burgess,** former director, resigned to become treasurer of the United Electric Coal Cos. of Chicago.

Gilbert Thiessen has been named manager of the laboratory section of the Koppers Co., Inc. central research department, Pittsburgh. He

Edward H. Rott

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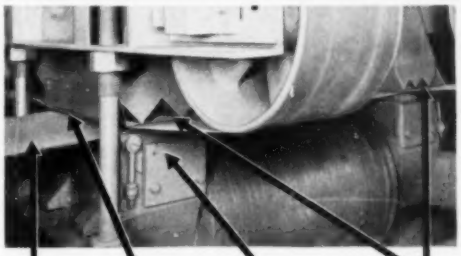

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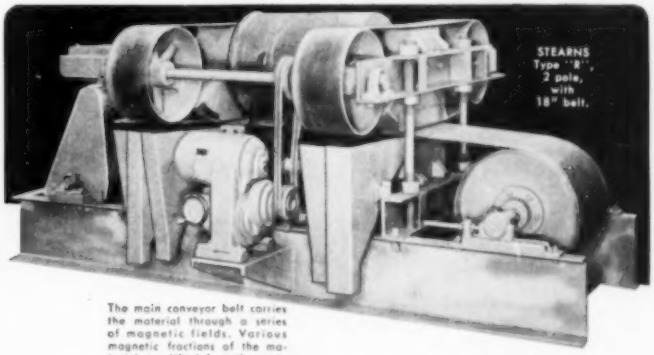

The main conveyor belt carries the material through a series of magnetic fields. Various magnetic fractions of the material are lifted from the main belt and carried away by the cross belt assemblies.

*Patent No. 2511484

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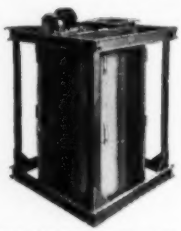

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was formerly technical adviser for the chemical division of the same concern.

Lauren A. Wright, associate mining geologist, Dept. of Natural Resources, Div. of Mines in Calif., was recently transferred from the San Francisco office to the Los Angeles office.

T. L. White is now employed by the Masara Consolidated Mines Corp., Escolta, Manila, Philippines. He was formerly connected with the La Luz Mines, Ltd., Nicaragua.

William P. Yant, director of research and development for Mine Safety Appliances Co., Pittsburgh, has been elected president for 1950-51 of the American Society of Safety Engineers.

Paul W. Zimmer resigned his position of geologist with the Chateaugay iron mining district, Republic Steel Corp., Lyon Mt., New York. He is now employed as geologist for the St. Joseph Lead Co., Balmat, N. Y.

—Obituaries—

Thomas Ellis Brown (Member 1916) died on June 24. Mr. Brown was born in 1889 at Yonkers, N. Y. and graduated from the School of Mines, Columbia University in 1912 with the degree of E.M. Following graduation he joined the Griscom Russell Co. and then the American Bridge Co. He became assistant consulting engineer to Thos. E. Brown in 1913. In later years he was connected with several concerns as a consulting engineer and in 1945 joined the Atlantic Beach Bridge Corp. In 1946 he became resident engineer and superintendent for the Nassau County Bridge Authority, Atlantic Beach Bridge, a position he held at the time of his death.

Melville F. Coolbaugh (Member 1918), president emeritus of the Colorado School of Mines, died suddenly on September 9 as the result of a heart attack. Born in Stroudsburg, Pa., he came to Adair County, Iowa as a child. He attended preparatory school at Wesleyan University in Lincoln, Neb., and was graduated from Colorado College, Colorado Springs in 1902, with a Bachelor of Science degree. He did graduate work at Columbia University, receiving an M.A. in chemistry and metallurgy in 1905, and also did special research work at MIT. During World War I he served in the chemical warfare division. From 1919 through 1925 he was director of metal research for Metals Exploration Co., Denver, and in 1925 was named president of the Colorado School of Mines. Prior to becoming president Dr. Coolbaugh

had taught chemistry at Colorado College, Columbia University, South Dakota School of Mines, Case School of Applied Science, Cleveland, and at Colorado Mines. During World War II he was regional representative for the government's engineering science and management war training program for

Melville F. Coolbaugh

Colorado, Wyoming, Utah, Idaho, and Montana. He held several state and federal appointments throughout his career when he served as secretary of the Colorado Geological Survey and was a member of the Colorado state planning commission from 1934 through 1945 and of the state mineral resources board. In 1946 he became president emeritus. After his retirement, Dr. Coolbaugh engaged in metallurgical research until the time of his death.

Dudley Stuart Dean (Member 1915) director of Calumet & Hecla Consolidated Copper Co. died September 26. Mr. Dean was born in Lake Village, N. H. in 1871 and educated at St. Paul's School, Concord and at Harvard University, where he graduated in 1891 with the degree of A.B. After his graduation he was employed by the Atchison, Topeka & Sante Fe Railroad in their operating department in Iowa, Missouri, and New Mexico. He served with Theodore Roosevelt's Rough Riders in the Spanish-American War. After the war Mr. Dean developed mining and land interests in Michigan, the Northwest and British Columbia. He was employed in gold dredging in the Boise Basin, Idaho, and prospecting and building gold dredge at Quesnelle River District of Cariboo district, B. C. During World War I he served as a captain in the Corps of Engineers. He served as chairman of the Keweenaw Land Assn., was president of the Ironwood Mines Corp., trustee and treasurer of the Newport Land Co., treasurer of the Porcupines Land Assn., and treasurer of the Barage Land Co.

Leo Ranney (Member 1926) mining engineer, died on September 15. Born in New Hartford, Iowa he graduated from Iowa State Teachers College in 1905 and spent two years in Alaska as a public school teacher. In 1911 he received a B.S. degree in geology from Northwestern University and then took engineering courses at Columbia and in 1914 began practice as a consulting engineer. During the first World War he was special assistant to the Army's Chief of Ordnance. In that capacity he studied methods of secondary recovery of crude oil. After the war he tried his method in Texas and filed patent applications. From 1920 to 1925 he worked as a consulting engineer in Texas and Oklahoma. In the latter year the Standard Oil Co. of N. J. bought his patents and made him president of the Ranney Oil Mining Co., a subsidiary. A superfluity of oil made his methods temporarily unnecessary and Mr. Ranney severed his connection with the company in 1930. He devised versatile horizontal techniques for obtaining water, oil, and coal gases from the ground. In 1934 he went to London and his water collector was successfully installed. He returned to the U. S. and leased his invention to several companies. In 1941 he

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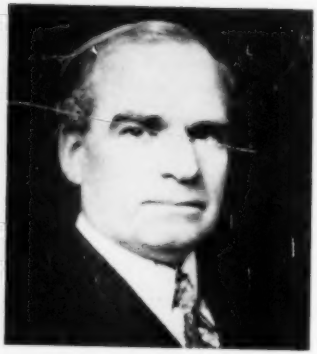

Leo Ranney

with the degree of B.S. in civil engineering and in 1908 received his M.S. In 1905 he began his service on the Cuyuna iron range of Minnesota at the time the range was being explored. In 1908 he was appointed geologist for the Northern Pacific Railway Co. and in 1925 was made manager of iron ore properties for the railway. He remained on the Cuyuna range during his entire career. He participated in many geologic surveys and studies in all states along the railroad's lines and had done considerable research work on the conversion of solid fuel to liquid fuel. He published many technical articles and historical treatises.

was vice-chairman of the Industrial Water Commission of Ohio and next year served as director of oil production for the Australian government.

Carl Zapffe (Member 1911) died on August 28. Mr. Zapffe was born in 1880 at Milwaukee and attended public schools there. Following graduation from high school he was employed by various companies and in 1903 entered the University of Wisconsin and graduated in 1907

Necrology

Date Elected	Name	Date of Death
1944	Fred A. Austin	March 1950
1940	Charles V. Averill	Aug. 28, 1950
1918	Paul W. Avery	Sept. 27, 1950
1939	Ralph S. Blitz	May 28, 1950
1942	Frank A. Bond	July 3, 1950
1940	W. E. Burlingame	Aug. 10, 1950
1942	E. C. Carlson	Sept. 9, 1950
1938	Willis H. Carrier	Oct. 7, 1950
1941	Cecil O. Dunlop	Aug. 10, 1950
1948	Charles A. Franks	Unknown
1917	Orr R. Hamilton	Apr. 24, 1950
1939	R. K. Matheson	Aug. 27, 1950
1943	Claude E. Needham	Oct. 15, 1950

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Total AIME membership on Sept. 30, 1950, was 16,737; in addition 4184 Student Associates were enrolled.

ADMISSION COMMITTEE

E. C. Meagher, Chairman; Albert J. Phillips, Vice-Chairman; George B. Corless, H. P. Croft, Lloyd C. Gibson, Ivan A. Given, F. W. Hanson, T. D. Jones, P. Malozemoff, Richard D. Mollison, and John Sherman.

Institute members are urged to review this list as soon as the issue is received and immediately to write the Secretary's Office, night message collect, if objection is offered to the admission of any applicant. Details of the objection should follow by air mail. The Institute desires to extend its privileges to every person to whom it can be of service but does not desire to admit persons unless they are qualified.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; AM, Associate Member; S, Student Associate.

Alabama

Anniston—Allen, John D. (J) (C/S—S-J).
Woodstock—Gunter, Howard C. (M).

Arizona

Ajo—Coope, Robert L. (J).
Ray—Hoyt, C. Leroy (M) (R. C/S—J-M).
Tucson—Littrell, Jimmie A. (J) (C/S—S-J).

California

San Diego—Parr, Albert F. Jr. (J) (C/S—S-J).
San Francisco—Whelon, James A. (J) (C/S—S-J).

Colorado

Boulder—Butler, Charles R. (J).
Golden—Bonardi-Bruce B. (J).
Lakewood—Tiernan, James B. (M).
Pueblo—Carter, David A. (M).

Connecticut

Westport—Hardin, Donald C. Jr. (J).

Illinois

Chicago—Anderson, Arne L. (J) (C/S—S-J).
Elmhurst—Huey, John J. (M).
Woodstock—Smith, Richard R. (J) (C/S—S-J).

Kansas

Pittsburgh—Kuperus, Cornelis (M).

Massachusetts

Boston—Needleman, Stanley (J).
Cheshire—Caldwell, Arthur B. (J) (C/S—S-J).
Weston—Linehan, Daniel (M).

Minnesota

Crosby—Appledorn, Conrad R. (J) (C/S—S-J).

Missouri

Bonne Terre—Kiser, John E. (M).
Rolla—Patrick, Donald J. (J) (C/S—S-J).

Montana

Butte—Bookey, Jack H. (J) (C/S—S-J).
Butte—Borden, James R. (A).
Butte—Kellogg, David A. (J) (R. C/S—S-J).
Butte—Norem, William C. (J) (C/S—S-J).
Butte—Stewart, Raymond M. (J) (C/S—S-J).
Butte—Williams, William D. (J) (C/S—S-J).

New Jersey

Engelwood—Brennan, Jerry E. (M).

New Mexico

Carlsbad—Reiter, Joseph S. (M).

New York
Lyon Mountain—Scott, Jack R. (M).
Newburgh—Dow, Melvin C. (M).
Tahawus—Strohl, John J. (M) (R, C/S—J-M).

Ohio
Cleveland—Johnson, Richard L. (M) (C/S—J-M).
Columbus—Wells, Donald R. (J) (C/S—S-J).
Euclid—Blackie, Benson D. (J) (C/S—S-J).
Mt. Pleasant—Sarff, Clifford L. (M).

Oregon
Hillsboro—Colegrove, Willard J. (M).
Hillsboro—Lewis, Albert A. (M) (R, C/S—J-M).

Pennsylvania
Bethlehem—Mack, Elmer L. (A).
Dupont—Kosik, Michael J. (M) (C/S—A-M).
Edwardsville—Rowlands, Richard (M).
Franklin—Calder, Alexander W. (J).
Pittsburgh—Ahl, Charles E. (M).
Pittsburgh—Purnell, Charles G. (A).
Plymouth—Evans, Isaac (A).
Scranton—Marcon, Frank B. (J).
Wanamie—Delestowicz, Stanley M. (M).
Wanamie—Waslofski, Walter B. (M).
Warrior Run—Richards, Edward (M).

Texas
El Paso—Lindberg, Richard E. (J) (C/S—S-J).
Houston—Coulter, James P. (M).

Utah
Murray—Gough, Ray F. (J) (C/S—S-J).

Virginia
Alexandria—Craig, Irving M. (J) (R, C/S—J-M).
Blacksburg—Clary, John H. (J) (C/S—S-J).
Blacksburg—Corrivean, Martial P. (J) (C/S—S-J).
Lynchburg—Roberts, Elford H. Jr. (J) (C/S—S-J).

Washington
Metaline Falls—Johnson, Edward L. (J) (C/S—S-J).

West Virginia
Bluefield—Williamson, Robert B. (M).
Holden—Turyn, John Jr. (J) (C/S—S-J).
Martinsburg—Lee, Richard W. (M).
Mount Hope—Bourland, Charles R. (M).
Oak Hill—Bennett, Wallace R. (M).
Rainelle—Weaver, Robert M. (J) (C/S—S-J).

Alaska
Sitka—Morgan, Mrs. Charlotte, T. (M).
Spenard—Erickson, Eugene P. (M) (R, C/S—S-M).

Bolivia
Llallagua—Merrow, Joseph H. Jr. (J) (C/S—S-J).

Brazil
Rio de Janeiro—Clemens, Wilbur M. (M).

Canada
Alberta—Stewart, John K. (J) (C/S—S-J).
Ontario—Mode, Donald H. (M).

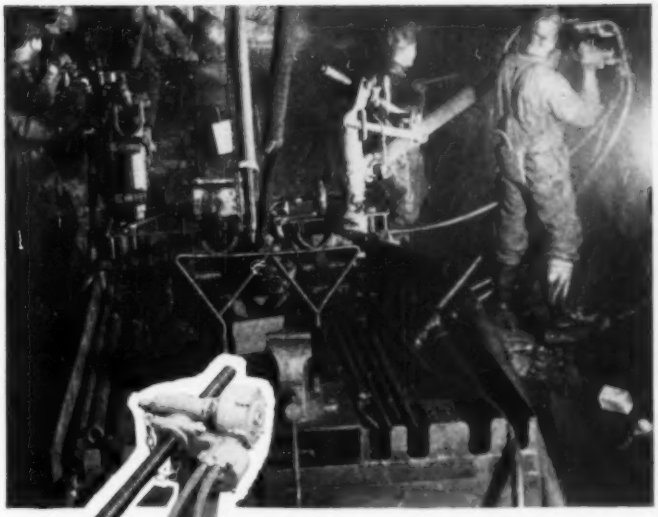
Chile
Portrerillos—Glahn, Reginald A. (J) (C/S—S-J).

Cuba
Pinar Del Rio—Jordan, Guy R. (M).

East Africa
Chunya Tanganyika Territory—Pickering, Bruce R. (A) (C, S—J-A).

Holland
Scheveningen—Levinson, Robert (J).

Mexico
Characas, SLP—Stringham, James R. (J) (C/S—S-J).
Chihuahua, Chile—Bacon, Francis G. (M) (R.).
San Luiz de la Paz, Gto.—Kohls, Albert E. (M).
Zacatecas—Manderfield, Charles Y. (J) (R, C/S—S-J).



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MOLY-COP COMPETITIVE BALL

The recovered test balls are shown in the top photograph. Clearly evident is the difference in rate of wear. The *competitive forged balls* wore 38% *faster*, based on average diameter loss. Note also the pronounced difference in retention of spherical shape.

Typical fractures of the two types of test balls are shown in the lower photograph. Note the fine, hard, martensitic grain structure of the Moly-Cop ball.

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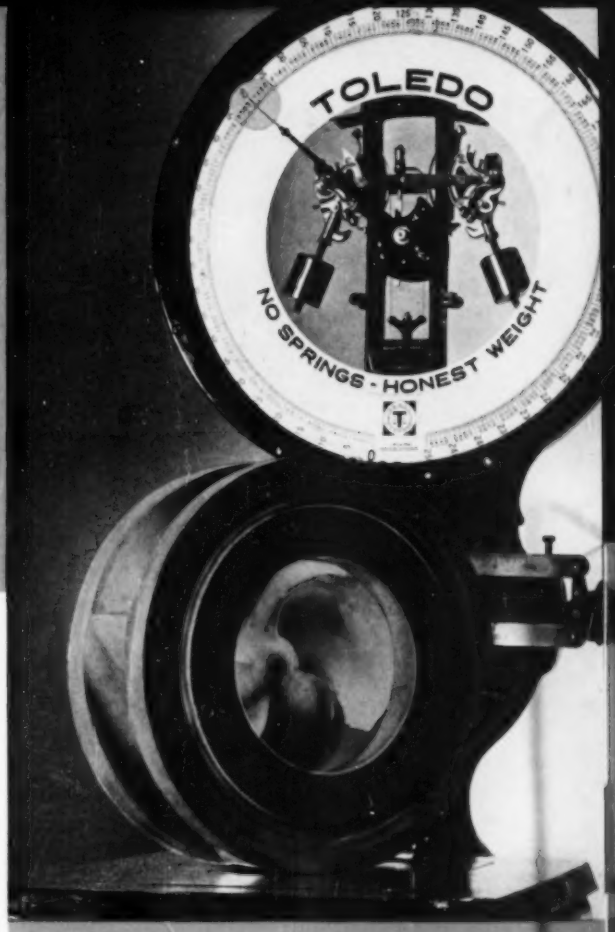

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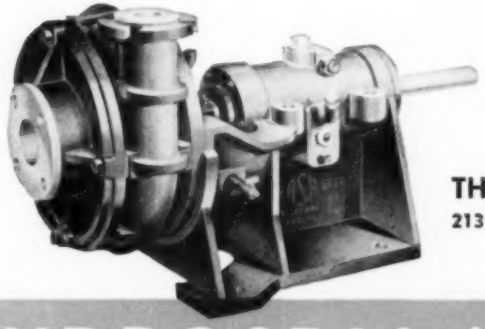

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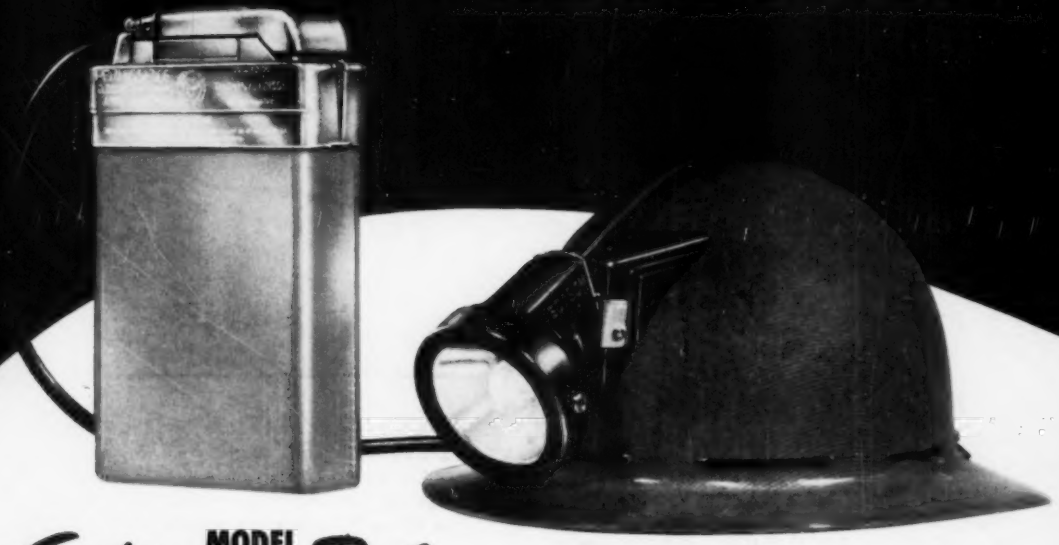

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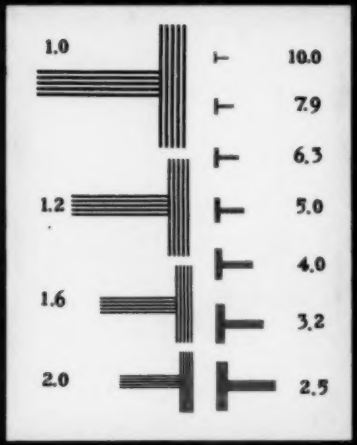

100 MILLIMETERS

INSTRUCTIONS Resolution is expressed in terms of the lines per millimeter recorded by a particular film under specified conditions. Numerals in chart indicate the number of lines per millimeter in adjacent "T-shaped" groupings.

In microfilming, it is necessary to determine the reduction ratio and multiply the number of lines in the chart by this value to find the number of lines recorded by the film. As an aid in determining the reduction ratio, the line above is 100 millimeters in length. Measuring this line in the film image and dividing the length into 100 gives the reduction ratio. Example: the line is 20 mm. long in the film image, and $100/20 = 5$.

Examine "T-shaped" line groupings in the film with microscope, and note the number adjacent to finest lines recorded sharply and distinctly. Multiply this number by the reduction factor to obtain resolving power in lines per millimeter. Example: 7.9 group of lines is clearly recorded while lines in the 10.0 group are not distinctly separated. Reduction ratio is 5, and $7.9 \times 5 = 39.5$ lines per millimeter recorded satisfactorily. $10.0 \times 5 = 50$ lines per millimeter which are not recorded satisfactorily. Under the particular conditions, maximum resolution is between 39.5 and 50 lines per millimeter.

Resolution, as measured on the film, is a test of the entire photographic system, including lens, exposure, processing, and other factors. These rarely utilize maximum resolution of the film. Vibrations during exposure, lack of critical focus, and exposures yielding very dense negatives are to be avoided.

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